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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

REVIEW OF STATE MINE INSPECTOR'S REPORTS AS THEY RELATE TO ACCIDENTS FROM FALLS OF ROOF

BY

J. W. PAUL

Circular No. 6110,
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REVIEW OF STATE MINE INSPECTORS' REPORTS AS THEY RELATE TO ACCIDENTS FROM FALLS OF ROOF [1]

By J. W. Paul[2]

A review of the published reports of the Mine Inspection Service of 19 States has been made to ascertain the character of the data relating to accidents from falls of roof and sides, to determine how the data will be helpful in a study of the problem, and to suggest methods for the prevention of such accidents. These 19 States produced 95 per cent of all of the coal in the United States during 1927, and falls of roof or sides in the mines of these States killed 1,067 men, a number equal to 54 per cent of all underground fatalities.

Mining officials and students interested in a study of the circumstances under which persons are killed by accidents in mining naturally look to official publications to supply the data needed for their analysis and for suggestions that may be helpful in outlining a campaign to curtail or prevent the occurrence of accidents from the major or dominating cause. Without a knowledge of the circumstances under which accidents occur, there can be no systematic or organized effort to combat the hazards that are responsible; but with a knowledge of the circumstances and conditions responsible for any class of accidents there should evolve a practical scheme for their elimination or curtailment.

The dissimilarity of the forms in which the reports are presented is most striking, as there is a lack of uniformity in many of the essential features relating to important data on accidents. However, many of the published reports, embody features which might be taken as models.

The main function of an Inspection Service report should be to give information concerning conditions affecting the efficiency of the service in protecting workmen against death or injury; this appears to be the primary purpose for which the service was created by the State.

This résumé of the State reports is not made in a spirit of criticism, but in the hope that future reports may be improved; reports should be presented with a degree of uniformity that will increase their value in a study of the circumstances and conditions contributing to accidents from falls of roof.

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2 Senior mining engineer, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

Table 1 - Summary of State Mine Inspectors' Reports, Giving Data Relating to Accidents from Falls of Roof and Coal

State	Date of issue of report	Pages in the report	Period covered by report year ending	Lives lost by falls[1]	Percent of all inside fatalities[1]	Is tabular data on accidents given: Fatal	Is tabular data on accidents given: Non-fatal	Is a special statement given in the text?	Are any remedies proposed in the text?	Are details of each fatal accident given?	Is place in mine given where the accident occurred?	Is mention made of condition of place or nearness of timber?	Is mention made of time of last inspection by an official?	Is mention made of timber regulations being neglected?	
Arkansas	-	59	June 30, 1925	N.G.	N.G.	No	No	No	No	No	No	No	No	No	
Colorado	4/1/27	80	Dec. 31, 1926	26	N.G.	Yes 44	Yes 1,716	Yes	No	Yes	Yes	No	No	No	Of 1,716 injuries underground, 593 were due to falls.
Illinois	-	235	June 30, 1924	Yes 55	Yes 49.1	Yes 112	3,352	Yes	No	Yes	No	No	No	No	Of 3,352 injuries underground, 919 were due to falls.
Indiana	-	29	Sept. 30, 1925	N.G.	N.G.	Yes 101	N.G.	No	No	No	No	No	No	No	
Iowa	5/ /24	-	Dec. 31, 1923	Yes 11	Yes	Yes	Yes	No	No	No	Yes	No	No	No	
Kansas	3/5/26	145	Dec. 31, 1925	Yes 5	N.G.	Yes	Yes	No	No	Yes	Yes	No	No	No	
Kentucky	6/30/26	290	Dec. 31, 1925	Yes 102	Yes 54.4	Yes	No	Yes	No	No	No	No	No	No	
Maryland	-	84	Dec. 31, 1925	Yes 8	N.G.	Yes	Yes	Yes	No	Yes	Partly	In some cases	No	In some cases	
Montana	-	3	June 30, 1926	Yes 4	N.G.	No	No	No	No	No	At face	No	No	No	
New Mexico	-	124	Oct. 31, 1924	Yes 10	N.G.	No	No	No	No	Yes	No	No	No	No	
North Dakota	12/31/26	80	Oct. 31, 1926	None	-	Yes 1	Yes 215	No	No	Yes	Yes	-	No	-	One fatal injury, resulting from breaking of cable. Of 215 non-fatal injuries, 24 were from falls.
Ohio	9/1/25	116	1924 and 1925	Yes	N.G.	Yes	Yes	No	No	No	No	No	No	No	
Pennsylvania Anthracite	5/29/25	99.	Dec. 31, 1921 and 1922 (2 years)	Yes 348	Yes 45.6	Yes	Yes	No	No	No	Yes	No	No	No	
Pennsylvania Bituminous	5/29/25	378	1921 and 1922 (2 years)	Yes 524	Yes 50.3	Yes	Yes	No	No	No	Yes	No	No	No	
Tennessee	-	135	Dec. 31, 1926	Yes 14	Yes 29	Yes	Yes	Yes	No	No	No	No	No	No	
Utah	-	-	2 years Dec. 31, 1926	Yes 33	Yes 76.6	No	No	No	No	Yes	No	No	No	No	
Virginia	-	6	Sept.30, 1926	Yes 36	Yes 73	Yes	Yes	No	No	No	No	No	No	No	
Washington	-	52	Dec. 31, 1919	Yes 5	Yes 35.7	Yes	Yes	Yes	Yes	Yes	Yes	Yes	No	No	
West Virginia [2]	-	324	Dec. 31, 1925	Yes 358	Yes 55.3	Yes	Yes	No	No	No	No	No	No	No	
Wyoming	-	45	Dec. 31, 1926	Yes 12	57.1	---	Yes	No	No	Yes	In some cases	No	No	No	

[1] N.G. = Not given.

[2] Period of 18 months.

Table 1 - Summary of State Mine Inspectors' Reports, Giving Data Relating to Accidents from Falls of Roof and Coal

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1 N.G. = Not given.

2 Period of 18 months.

The fatality accidents underground for any one year occur in mines which produce practically half the nation's coal, and in the mines producing the remainder of the coal tonnage for the same period there are no fatalities. This condition prevails year after year, but the mines that are free of fatalities during one year may join the group having fatalities during the next year. The list of mines that make up these two groups is constantly changing.

The reports reviewed are not all for the same period; owing to the lateness of their appearance, it was necessary to go back as far as 1919 for one of them. The general outline of the reports from any given State for one year is similar to previous and subsequent issues; therefore, the reports that were selected were typical of previous issues.

SUMMARY OF REPORTS TABULATED

Table 1 is a summary and comparison of State inspectors reports as they relate to accidents from falls of roof and coal. It also indicates those that are deficient in data either in tabular form or in the text. It is believed that the information called for by the headings of this table when compiled for each State would be a great aid in analyzing the primary and ultimate cause of accidents from falls, and also in devising methods for the prevention of such accidents.

SPECIAL STATEMENT IN TEXT

Most reports begin with an introduction to and summary of the contents; and as the prevention of accidents is the main reason for issuing the State mine inspector's reports, it would be natural to expect to find a discussion of the accidents in the main text. However, of the 20 reports, covering 19 States, 14 make no special mention of the accidents.

REMEDIES PROPOSED

One report presents a brief discussion of a proposed remedy for all accidents which will place the responsibility definitely for each accident. In none of the other 19 reports are any proposed remedies discussed.

DETAILS OF ACCIDENTS

Details of fatal accidents, strangely, are given little prominence in many of the reports. Eleven reports give no details, and 9 reports give details of each fatality. Four reports do not give a tabular statement of accidents by causes.

WHERE ACCIDENTS OCCURRED

Seven reports give the place in the mine where the accident occurred; two others give this information only partly; 11 reports do not furnish this information.

CONDITION OF PLACE AND NEARNESS OF TIMBER

In a study of the probable cause of falls of roof it is always of prime importance to know the physical condition of the vicinity where the accident occurred and the position of any timber that had been placed for protection against such falls. In only two reports is this information given, and in one of them it is given only in part.

PRIOR INSPECTION BY AN OFFICIAL

Supervision (direction) and discipline probably constitute the main force behind a safety program for the prevention of accidents, so that when an accident does occur it is of much importance to determine wherein supervision or disciplin may have been remiss. This involves ascertaining when the scene of the accident was last visited by an official prior to the occurrence of the accident, and the nature of any instructions that may have been given. None of the reports reviewe gives this information.

NEGLECT OF TIMBER REGULATIONS

Only one report mentions the condition of the timbering at the scene of fatal accidents by falls of the roof, but it does not give this information for all fatal accidents.

DISCUSSION

Much, if not all, of the information on fatal accidents from falls is believed to constitute a part of the office records of the State Mine Inspector. The incorporation of this data in the published reports may have been omitted either as economy in the cost of printing or on the assumption that the information is of value only to the inspection service. However, it is found that one report gives all but two of the main headings named above; another report gives all but three, whereas one report gives none.

SUGGESTED OUTLINE FOR REPORT

In the interest of uniformity or standardization of State reports the following outline is presented for consideration by those who have in charge the assembly and compilation for publication of data on accidents.

ACCIDENTS

1. Fatal, inside: Discussion by causes, number by each cause; ratio per million tons of product, ratio per thousand 300 shifts, ratio per 1,000 underground employees.

2. Fatal, outside: Same as for 1, inside.

3. Give tabulation by Bureau of Mines form. (See form A).

4. Nonfatal, inside: Discuss by causes and number by causes permanent total disability, permanent partial disability, temporary disability, giving nature of injury and total lost time.

PROPOSED REMEDIES, DISCUSSION OF

The data developed by the use of the special forms will enable such an analysis to be made as will admit of intelligent discussion of causes of accidents and suggest remedies which may lead to material reduction of accidents from falls. At least the use of such forms and the analysis of the data will indicate the point where it will be profitable to make an attack.

With the view of unifying and simplifying the presentation of the data on accidents, the 9 forms following have been prepared and are submitted for consideration by those who direct the preparation of publications on mine accidents.

FORM A

Coal mine fatalities from all causes

Cause of accident	No. killed	Per cent
UNDERGROUND		
Falls of roof (coal, rock, etc.):		
Ordinary disaster a		
Major disaster b		
Falls of face or pillar coal		
Mine cars and locomotives:		
Ordinary disaster		
Major disaster		
Explosions of gas or coal-dust:		
Ordinary disaster		
Major disaster		
Explosives:		
Ordinary disaster		
Major disaster		
Suffocation from mine gases:		
Ordinary disaster		
Major disaster		
Electricity		
Animals		
Mining machines		
Mine fires (burned, suffocated, etc.):		
Ordinary disaster		
Major disaster		
Other causes:		
Ordinary disaster		
Major disaster		
Total		
Ordinary disaster		
Major disaster.		
SHAFT		
Falling down shafts or slopes		
Objects falling down shafts or slopes		
Cage, skip, or bucket:		
Ordinary disaster		
Major disaster		
Other causes		
Total		
Ordinary disaster		
Major disaster		

a An accident in which less than five were killed.
b An accident in which five or more were killed.

Coal mine fatalities from all causes - Continued

	No. killed	Per cent
SURFACE		
Mine cars and mine locomotives		
Electricity		
Machinery		
Boiler explosions or bursting steam pipes		
Railway cars and locomotives		
Other causes:		
Ordinary disaster		
Major disaster		
Total		
Ordinary disaster		
Major disaster		
Grand total		
Ordinary disaster		
Major disaster		

Total days mine was in operation during year shift of ____________ hours	
Surface employees	
Total employed underground and on surface	
Total days tipples operated during year shift of ____________ hours	

FORM B

Occupation and nationality of persons employed underground and number killed and injured by falls

OCCUPATION UNDERGROUND	NATIONALITY														
	American			Italian									Totals		
	Total	Killed	Injured	Total	Killed	Injured	Total	Killed	Injured	Total	Killed	Injured	Total	Killed	Injured
Workers at the "face":															
Pick miners and pick loaders or laborers															
Machine runners, cutters, helpers, and scrapers															
Machine miners and loaders															
Operators of mechanical loaders, scrapers, or conveyors															
Shot firers and runners															
Fire bosses, mine examiners, fire examiners, gasmen															
Face bosses															
Haulage workers:															
Drivers and boss drivers															
Motormen															
Motormen's assistants , brakemen, trip riders, patcher switchmen, snappers, runners, gripmen															
Doorboys or trappers															
Cagers															
Spraggers, car-couplers, switch tenders, greasers															
Others underground:															
Mine foremen, managers, and pit bosses															
Assistant foreman, managers, pit bosses															
Stablemen															
Trackmen, roadmen, tracklayers															
Pumpmen, pipemen, and helpers															
Timbermen, bratticemen, rockmen, and helpers															
Wiremen, electricians, and helpers															
Hoistmen (stationary engineers)															
All others															
TOTALS															

FORM C

Place in mine where fatal accident occurred from falls of roof

Location	No. killed
In room at or near face	
In room along haulage track	
In room at or near its entrance	
In entry at or near face	
In entry along haulage road	
In pillar work	
In other locations	

FORM D

Occupation of person at the time of fatal accident from falls of roof

Occupation	
Loading coal	
Undercutting coal	
Setting timber	
Removing timber	
Preparing shot	
Laying track	
Testing roof	
Taking down loose roof	
Traveling to or from working place	
Returning after blasting	

FORM E

Conditions of working places where fatal accidents occurred from falls of roof or sides

No. of persons killed working alone	
No. of persons killed working with another person	
No. of persons killed working in narrow place, 9 to 14 feet	
No. of persons killed working in wide places, 15 to 20 feet	
No. of persons killed working in wide places, 20 to 30 feet	
No. of persons killed working in longwall face	
No. of persons killed extracting pillars	
No. killed where timber was within 3 feet of face	
No. killed where timber was within 4 feet of face	
No. killed where timber was within 5 feet of face	
No. killed where timber was within 6 feet of face	
No. killed where timber was within 8 feet of face	
No. killed where timber was within 10 feet of face	
No. killed where timber was within 12 feet of face	
No. killed where timber was over 12 feet of face	
No. killed where no timber was used	
Total	

[illegible]

FORM F

Data on timbering where fatal accidents occurred
from falls of roof

Number of persons killed where no regular

system of timbering was observed ------------------------

Number killed where there was a regular

system of timbering -------------------------------------

Number killed through neglect in complying

with system of timbering ---------------------------------

FORM G

Hour of day at which fatal accidents occurred
from falls of roof

A.M.	No. killed	Percentage of grand total	P.M.	No. killed	Percentage of grand total
Midnight to 6 A.M.			12:01 to 1		
6:01 to 7			1:01 to 2		
7:01 to 8			2:01 to 3		
8:01 to 9			3:01 to 4		
9:01 to 10			4:01 to 5		
10:01 to 11			5:01 to 6		
11:01 to 12			6:01 to midnight		
			Total		100.0

FORM H

Number of lives lost by falls of roof or sides with respect to the interval elapsing between occurrence of accident and time when the place was visited by an official

Interval between official visit and accident	No. killed
During interval of less than 30 minutes	
During interval over 30 minutes and less than 1 hour	
During interval over 1 hour and less than 2 hours	
During interval over 2 hours	
During interval of unknown duration	
Total killed by falls	

FORM I.

Underground experience of persons killed by falls of roof

Length of experience in coal mines	No. killed
Less than 30 days	
From 30 to 60 days	
Over 2 months, less than 4 months	
Over 4 months, less than 6 months	
Over 6 months, less than 8 months	
Over 8 months, less than 10 months	
Over 10 months, less than 1 year	
Over 1 year, less than 2 years	
Over 2 years, less than 3 years	
Over 3 years, less than 4 years	
Over 4 years, less than 5 years	
Over 5 years, less than 6 years	
Between 6 and 10 years	
Between 10 and 15 years	
Between 15 and 20 years	
Between 20 and 25 years	
Over 25 years	
Unknown	
Total	

Circular 6111 April, 1929

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

MINING LAWS OF CHINA

BY

JOHN W. FREY

Circular No. 6111
April, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

VI. MINING LAWS OF CHINA[1]

By John W. Frey[2]

PREFATORY NOTE

This paper presents one of a series of digests of foreign mining legislation and court decisions which is being prepared in advance of a general report relative to the rights of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the laws of China was prepared with the aid of a questionnaire submitted through the courtesy of the division of commercial laws of the Bureau of Foreign and Domestic Commerce to Wilbur K. White, Assistant Trade Commissioner, Peking, China, who had the assistance of Dr. W. H. Wong, Director of the Geological Survey of China, and Mr. Edwin W. Mills of the American community.

SYNOPSIS OF LAW

In theory the State owns all minerals, with the possible exception of certain nonmetallic minerals. These minerals, known as Class III minerals and consisting mainly of building materials, may be exploited by the surface owner, who is also privileged to lease the property to others for exploitation, subject, however, to obtaining the approval of the higher local administrative authorities. (Articles of the Chinese Mining Enterprise Regulations promulgated on March 11, 1914, Art. 11.) The working of salt and petroleum are considered State monopolies, although with respect to petroleum the law is not strictly applied.

The minerals of China are divided into three classes: <u>Class I</u> includes gold, silver, copper, iron, tin, lead, antimony, nickel, cobalt, manganese, zinc, aluminum, arsenic, mercury, bismuth, platinum, iridium, molybdenum, chromium, uranium, coal, and precious stones; <u>Class II</u> includes rock crystal, asbestos, mica, corundum, emery, gypsum, apatite, barites, nitrates, sulphur, pyrites, borax, fluorspar, marble, feldspar, talc, graphite, peat, amber, asphalt, bitumen, pumice, meerschaum, kaolin, diatomaceous earth, tripolite, magnesium earth (magnesite?), fuller's earth and "stones used to make pigments"; <u>Class III</u> includes slate, limestone, sandstone, granite, porphyry, dolomite, lime, marl, fire clay and other useful stones quarried for architectural and manufacturing purposes.

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2 Associate mineral economist, U. S. Bureau of Mines, Washington, D. C.

The ownership of the surface and of the subsoil are separated. With regard to the minerals of Class I, those persons, whether surface owners or not, who shall first petition for claims shall have the prior right to secure such claim. (Ref. cit. Art. 9.) Accordingly, fee simple ownership is not possible.

ALIEN RESTRICTIONS

Americans are not permitted to explore, own, and operate mines on the same terms as Chinese citizens, as mining rights may be acquired only by citizens of the Republic of China or by individuals who have lawfully acquired Chinese citizenship. (Ref. cit. Art. 3.) However, citizens of the United States or of other treaty nations may join with Chinese citizens in acquiring mining rights subject to the mining regulations and certain other laws connected therewith. (Ref. cit. Art. 4.)

Foreigners are not allowed to hold more than 50 per cent of the total number of shares, and each foreigner is required to present to the Administrator of Agriculture and Commerce or to the Director of the Mining Supervision Office a document issued by a diplomatic or consular officer of his own country, certifying that he is willing to adhere to the mining regulations (idem). In doing joint business with citizens of the Chinese Republic it is necessary to incorporate under the laws of China; consequently, a study of Chinese company law must be made.

American are not permitted to explore freely. No minerals of Classes I, II, and III, or even the waste dumps of abandoned mines, can be prospected or mined except by previously obtaining the approval of the Minister of Agriculture and Commerce or the Director of the Mining Supervision Office. (Ref. cit. Art.8).

PROSPECTING

The mineral or minerals to be sought in prospecting must be specified in the petition for a mining permit. The petition must be accompanied by maps and explanatory remarks. The Director shall, if he deems it necessary, instruct local officials to investigate, or he himself may appoint a deputy to make a personal inspection of, the area applied for. (Ref. cit. Art. 25).

The prospector is not free to prospect within certain excepted areas such as the following:

1. Lands within one li[3] from the boundaries of the tombs of the Ancient Sages, emperors, or kings.

2. Lands of importance to fortifications, strategic points, or arsenal and naval bases -- unless approval has been obtained from the controlling government offices concerned.

3. Areas within one li of commercial centers or trading markets unless consent has been obtained from the controlling government offices concerned.

3 About one-third mile.

4. Areas within 400 ch'ich, or Chinese feet[4], from the sites of official or public buildings, public parks, famous ancient monuments, public thoroughfares, railways, and important waterways or water systems, etc., unless approval has been obtained from the controlling government offices concerned, the owner, or other persons interested (Ref. cit. Art. 13).

The owner of the land has the right to prevent prospecting unless rent or compensation is paid for the land. (Ref. cit. Art. 59). The use by claim-holder of land belonging to other persons for mining operations is permissible, but when the land of others is utilized the approval of the Director of the Mining Supervision Office must first be obtained. At the same time a petition with the working plan, drawing, and a description must be submitted to the Director for his approval. After granting such permission the Director immediately notifies the landowner or person concerned, but that does not relieve the claim-holder from his obligation to consult with the landlord. Should the land be government property, the claim-holder may apply for it to the officer in charge. (Ref. cit. Art. 57 and 58). In case the use of the land is to extend over a period longer than three years, or if in consequence of such use its character shall undergo change, the holder of the mining right may negotiate with the landowner or the landowner may demand a lump sum as compensation according to the market value of the land. During the period of utilization of the land the rights of ownership are vested in the claim-holder, and the rights of the landowner are temporarily suspended. (Ref. cit. Arts. 60, 67, 68).

The prospecting area is limited in size. The minimum area for coal mining shall be not more than about 41 acres (270 mow) and the maximum area not more than 820 acres (10 square li). The area of other mines shall be over 7.6 acres (50 mow) and not over 410 acres (5 square li). (Ref. cit. Art. 16). The prospecting rights are limited to a period of two years (Ref. cit. Art. 26), and payment must be made of the appropriate fees as listed herein on a later page.

Generally speaking, the prospecting license does not require that a certain amount of work must be done to keep it in force. (e. g. assessment work). However, a time limit may be imposed within which the petitioner must file an application for operating rights in said area, whenever in the opinion of the Minister of Agriculture and Commerce or the Director of the Mining Supervision Office, the area to which the prospecting rights relate actually might be operated as a mine. In case the petitioner fails to petition within this imposed time limit, the Minister or the Director may allow other persons to apply. (Ref. cit. Art. 32).

MINING CONCESSIONS

In granting mining claims, the issuing officials have rather wide discretionary powers. Normally the Director of the Mining Supervision Office petitions for permission from the Minister of Agriculture and Commerce. As in the case of prospecting licenses the Minister, if he considers it necessary, may instruct the Director or appoint a special deputy to make personal investigations. (Ref. cit. Art. 28). If it appears that the situation and shape of the mining

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4 Equivalent to 1.04987 English feet. (According to the Statesman's Yearbook, the ch'ich varies in different localities from 9 to 16 inches).

area do not agree with the situation and shape of the mineral deposit, thereby causing prejudice to profitable working of the mine, the Minister or the Director may impose a time limit within which an amended petition must be submitted. In case the amended petition is not filed within the time limit imposed, the original petition shall be cancelled. (Ref. cit. Art. 33). It is the prerogative of the Director of the Mining Supervision Office to make investigations regarding the area applied for as a mining claim, and should the place be unsuitable for such an enterprise or should the enterprise be injurious to the public interest the petition may be rejected. (Ref. cit. Art. 34). Prospecting claims can not be increased, decreased, amalgamated, divided, or altered without the approval of the Director of the Mining Supervision Office. (Ref. cit. Art. 43). Also the holder of the mining claim is restricted by having to submit to the Director for his approval, any plans for proposed alterations in working the claim. (Ref. cit. Art. 44). The issuing officials have the right to refuse to grant a permit, but they rarely do so.

Since the promulgation of the mining laws of 1914, only mining areas within a definite area have been granted. Previously special charters were granted for such foreign and semiforeign concessions as the Fushun, Penchihu, Kailan and Peking Syndicate; and for the Pinghsiang, Tayeh, Shiukoushan, and other Chinese mines. Leases or concessions of this sort are no longer granted and at the present time all new grants for mining are handled under the claim system.

The rights to a mining claim are for an indefinite period, but they may be cancelled for any one of the following causes:

1. If without reasonable cause operations have not begun within one year after registration or after operations have commenced, work is suspended for one year or more.

2. If the mining operations are injurious to the public interests.

3. If there has been a failure to comply with the mining police regulation

4. If the approved plans and descriptions of work have not been complied with.

5. If the mining taxes have not been paid up to date.

6. If the sanction was given in error.

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

Schedule of Fees for Petitions, in local currency.[5]

To accompany petition regarding:		Each
1.	Prospecting permit	$20.00
2.	Changes of Area:	
	Increases or increases and decreases combined	12.00
	Decreases	2.00
3.	Alteration of prospecting Area:	
	Increases or increases and decreases combined	12.00
	Decreases	2.00
	Other alteration	2.00
4.	Changes of Petitioner (Claim Owner) for prospecting:	
	Due to inheritance	2.00
	Due to other causes	10.00
5.	Working of Mine (Operating Permits)	30.00
6.	Changes as to ground under petition for operation:	
	Increases or increases and decreases combined	20.00
	Decreases	2.00
7.	Change of Mining Area:	
	Increases or increases and decreases combined	20.00
	Decreases	2.00
	Rectification of Area	12.00
	Alteration of Area	2.00
8.	Amalgamation or division of mining areas	12.00
9.	Separation and amalgamation of mining areas	20.00
10.	Change of petitioners for mining operations:	
	Due to succession	2.00
	Due to other reasons	20.00
11.	Withdrawal of a partner	2.00
12.	Correction of name of mineral	4.00
13.	Inspection according to Art. 50 of the Mining Regulations	20.00
14.	Request for survey or inspection according to Art. 53 of the Mining Regulations	4.00
15.	Request for removal of obstacles according to provisions of Art. 54 of the Mining Regulations	6.00
16.	Request for use of other person's land-property according to provisions of Art. 57 of the Mining Regulations	10.00
17.	Request for final decision of a case	10.00

5 From Art. 37 of the regulations promulgated March 31, 1914.

MINING TAXES

Mining taxes are of two sorts -- the mining-area tax and the mineral-production tax. The rate of taxation on mining areas is as follows:

1. For mining areas in which minerals included under Class I are worked, the annual tax per <u>mow</u> is 30 cents (local currency).

2. For minerals of Class II the annual tax per <u>mow</u> is 15 cents (local currency).

The mineral-production tax is as follows:

For minerals in Class I, 1.5 per cent of the market price at the place of production; for minerals Class II, 1 per cent of the market price at the place of production; for minerals in Class III, no taxes of any sort are levied. If the claim is being prospected the above tax shall be reckoned at 0.05 cents per <u>mow</u>.

SUMMARY

The most important points in the present mining regulations are as follows:

1. The recognition of mining rights apart from the ownership of surface rights.

2. The adoption of the claim system granting rights of priority on application with the proper maps and descriptions.

3. A mining tax of $1\frac{1}{2}$ per cent on the precious metals and coal.

4. The limitation of foreign capital, which must not exceed 50 per cent.

5. Foreigners must be represented by Chinese in dealing with the authorities.

6. Mining rights may be forfeited for any one of six reasons: Cessation of work for a year; injury to public interests; noncompliance with police regulations; mistaken sanction to operations; failure to work in strict accordance with approved plans; or nonpayment of taxes.

The new laws of the Republic may be more favorable to foreigners. According to Dr. Wong Wen-hao, Director of the Geological Survey of China, a commission has been appointed by the President of the Nationalist Government at Nankin for the purpose of drafting new mining laws and regulations with the express purpose of encouraging a healthy development of the mining industry in China and to make conditions attractive for the introduction of foreign capital.

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Circular 6112 April, 1929

INFORMATION CIRCULAR

Department of Commerce -- Bureau of Mines

WHAT DO WE KNOW ABOUT THE EXPLOSIBILITY OF COAL DUST IN MINES

BY

H. P. Greenwald

Circular No. 6112,
April, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

WHAT DO WE KNOW ABOUT THE EXPLOSIBILITY OF COAL DUST IN MINES? [1]

By H. P. Greenwald[2]

INTRODUCTION

The Bureau of Mines has been conducting experiments on the explosibility of coal-dust in the experimental mine for more than 17 years. The results have been published from time to time[3], and a forthcoming technical paper[4] will summarize the results and discuss them in some detail. The present paper will be limited to a statement of the more important facts that these researches have discovered.

EXPLOSIBILITY OF COAL-DUST AND MEANS OF PREVENTING EXPLOSIONS

It is known that coal-dust is explosive only when raised as a cloud in air, and that a certain minimum concentration in the cloud must be obtained or exceeded before flame will pass through the cloud. There is almost everywhere in a coal mine ample dust deposits from which if untreated the coal dust clouds necessary to obtain a coal dust explosion can be formed, and the means of forming and igniting them are present only too frequently.

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1 The Bureau of Mines will welcome reprinting of this article but requests that the following footnote acknowledgment be used: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"
2 Physicist, experimental mine section, U. S. Bureau of Mines.
3 Rice, G. S., Jones, L. M., Clement, J. K., and Egy, W. L., First Series of Coal-Dust Explosion Tests in the Experimental Mine: Bull. 56, Bureau of Mines, 1913, 115 pp.
Rice, G. S., Jones, L. M., Egy, W. L., and Greenwald, H. P., Coal-Dust Explosion Tests in the Experimental Mine, 1913 to 1918, Inclusive: Bull. 167, Bureau of Mines, 1922, 639 pp.
Rice, G. S., Paul, J. W., and Greenwald, H. P., Coal-Dust Explosion Tests in the Experimental Mines, 1919 to 1924, Inclusive: Bull. 268, Bureau of Mines, 1927, 176 pp.
Rice, G. S., Greenwald, H. P., and Howarth, H. C., Coal-Dust Explosion Tests in the Experimental Mine, January, 1925 to March, 1926 (in preparation).
4 Rice, G. S., and Greenwald, H. P., Factors Affecting the Explosibility of Dust in Coal Mines: A Summary of Results of Investigations in the Experimental Mine (in preparation).

Before examining the explosibility of coal-dust some measure must be adopted for the property which is so named. It has been found best to consider explosibility proportional to the amount of incombustible material required in a coal-dust rock-dust mixture to prevent propagation of flame through that mixture, provided that it is dry enough to form a dust cloud readily. This rules out mixtures containing water, which will be considered later. After adopting this standard it is necessary to select a standard rock-dust or to investigate the relative effectiveness of several rock-dusts. The latter course has been followed to some extent with finely sized dry dust, and the following factors have been found: (1) The effectiveness of rock-dust is independent of chemical composition, provided that there is not more than 2 per cent combustible material in it. Thus, dusts made from limestone, soapstone, adobe, shale, and gypsum are all equally effective. (2) Approximately equal weights of different rock-dusts are required, irrespective of their specific gravities. The effectiveness of diatomite-dust has been compared with that of shale-dust in gallery tests, and it has been found that only slightly less weight of the former was required, despite the fact that its volume per unit weight was 4 to 6 times that of the shale. (3) The sige of the rock-dust is more important than either its composition or specific gravity, provided of course that it contains no combustible material.

Two sizes of rock-dust were used in early work at the experimental mine. The first was called pulverized dust, of which about 98 per cent would pass a 200-mesh sieve; the second was called 20-mesh dust, of which about 95 per cent would pass 20-mesh sieve, and 35 to 40 per cent would pass a 200-mesh sieve. It was found that the finer dust was the more effective, but that the difference could be offset by using 3 to 5 per cent more of the coarser dust. When rock-dusting specifications were drawn, it was stated that the dust used should be ground so that all would pass a 20-mesh sieve and 50 per cent would pass a 200-mesh sieve. This was considered to be the best compromise between effectiveness and cost, which mounts rapidly with fine grinding. This specification was also adopted by the American Engineering Standards Committee and it is the British standard size for stone-dust as well.

FACTORS INFLUENCING EXPLOSIBILITY OF COAL-DUST

Experimental evidence may be said to show that the explosibility of a given coal-dust is not constant even when present in the proper amount in a given mine passageway. The explosibility is influenced by the following factors:

1. Size of the dust.
2. Composition of the dust.
3. Presence of fire damp in the air.
4. Quantity of coal dust present
5. Manner in which the dust is distributed.
6. Characteristics of the igniting source.
7. Surrounding conditions.

1. Size of dust.- Mine dust is defined as all material which will pass a 20-mesh sieve. This specification was first made in 1909 following certain gallery tests and was later supported by tests in the experimental mine in which the addition of material coarser than 20-mesh did not change the quantity of rock-dust required to prevent propagation of flame through coal-dust finer than 20 mesh. It is assumed that in general the coarser material does not take part in an explosion when an ample amount of finer material is present. An exception might occur in explosions developing so much violence as to abrade coal from the ribs and cause a general degradation in the size of the dust. The question of the size of dust, however, can not be dismissed with the statement that all must pass a 20-mesh sieve. Further subdivision has been found necessary, and dusts are classified according to the amount of minus 200-mesh material which they contain. Tests have been made in the experimental mine with dusts having 10, 20, 40, and 85 per cent through 200 mesh. The last is commonly called pulverized dust. The dusts found in many mines have been examined in connection with explosion-hazard investigations, and it has been found that mine dusts usually have an average of 15 to 25 per cent of dust that will pass a 200-mesh sieve. Road dusts are coarser and rib dusts much finer than this average. The road dusts greatly outweigh the rib dusts in quantity. As a result of this, dust having 20 per cent through 200 mesh has been most used in tests of coals received from producing mines in various parts of the country.

2. Composition of the dust.- In a broad sense composition should cover variations in the moisture, volatile matter, and ash content of the dust. However, the moisture and ash of coal can not be distinguished from the moisture and ash of admixed inert material either in a mine dust or in a mixture prepared for test in the experimental mine. It is customary to group the incombustible portion of the coal with admixed combustible, and although the effect of the two is probably not proportional to their relative weights, the quantity of incombustible in the coal itself is relatively so small in most cases that any difference in action may be ignored. Study of the effect of varying composition then resolved itself into a study of the effect of variations in volatile matter content, and these are best expressed on a moisture and ash-free basis. They are the same as the ratio of volatile matter to total combustible, which varies from 0.05 for anthracites to 0.50 for subbituminous coals. In order to determine the matter experimentally, it was necessary to specify the size of dust used and the method under which the dusts should be tested. Consideration is given here only to the relative explosibility of dust having 20 per cent through 200 mesh when subjected to the standard propagation test in the experimental mine.

Details of this test method are given in Bureau of Mines Bulletin 268, page 36. Briefly, the dust mixture under test is subjected to the explosion developed by 100 pounds of pulverized Pittsburgh coal-dust when ignited by a blowout shot of 4 pounds of black blasting powder. Dusts of the size specified, prepared from 29 different coals, have been subjected to this test. The conclusions drawn are summarized in the following table which gives the proportion of incombustible material required in mixtures of coal and rock dust to prevent propagation of an explosion through them under the test conditions.

<u>Incombustible required to prevent propagation of an explosion through 20 per cent 200-mesh dusts of various coals</u>

Volatile ratio of coal	Incombustible required, per cent.	Volatile ratio of coal	Incombustible required, per cent
0.14	14	0.23	60
0.15	20	0.25	61
0.16	25	0.30	61
0.17	31	0.35	61
0.18	36	0.40	61
0.19	42	0.43	63
0.20	47	0.46	66
0.21	53	0.49	69
0.22	58		

Explosibility increases rapidly between ratios 0.14 and 0.25, remains constant between ratios 0.25 and 0.40, and then increases slowly. As most of the coals now being mined in the eastern part of the United States have volatile ratios lying between 0.25 and 0.40, it follows that 61 per cent incombustible is required in a majority of the mines when no gas is present.

3. <u>Fire damp in the air</u>.- The presence of fire damp in air increases the explosibility of a coal-dust cloud raised in that air. It has been found from many experiments that there is a definite rule connecting this increase in explosibility with the explosibility of the dust when no fire damp is present and the lower limit of explosibility of fire damp air mixtures with no dust present. The limit of explosibility of the dust with no fire damp present must be known to apply the rule which reads: Subtract the no-gas limit from 100 and divide by 5. The result is the percentage of incombustible required to offset each 1 per cent of fire damp in the air current. Thus, if a coal requires 60 per cent incombustible with no fire damp present, $\frac{100-60}{5}$ = 8 per cent additional incombustible is needed for each 1 per cent of fire damp in the air current. If the requirement was only 30 per cent with no fire damp, then each 1 per cent would require $\frac{100-30}{5}$ = 14 per cent additional incombustible material. Under this rule 100 per cent incombustible must be present with 5 per cent fire damp, which is approximately the lower limit of inflammability of fire damp-air mixtures when strong sources of ignition are used.

4. <u>Quantity of Coal Dust</u>.- We do not know precisely how small an amount of coal dust already in suspension in air would propagate an explosion, but as little as 4.8 ounces of coal dust per linear foot of entry, when distributed on cross-shelves and side shelves, has done so. To produce maximum violent effects, the quantity of coal dust per unit of entry or room space varies with the size of dust particles and the stage of the explosion at the particular place in the mine. An excess of coal dust will tend to absorb heat and lessen violence but will not prevent flame from propagating with disastrous results.

5. Distribution of dust.- Mine dust is usually found on every point in the entry perimeter, on the floor, ribs, timbers, and adhering to the roof. When it is dislodged from elevated surfaces, gravity aids in forming a cloud; when raised from the floor, gravity opposes the formation of a dust cloud. In this manner the explosibility of mine dust is influenced by its original place along the perimeter of the entry. The effect on explosibility was demonstrated in the early work in the experimental mine. At first, dust was placed only on longitudinal side shelves and on the floor. Later, cross-shelves were installed near the roof at 10-foot intervals and part of the dust was placed on them. Under these conditions the amount of rock-dust required to prevent propagation of an explosion was increased 10 to 15 per cent. In operating mines the problem is complicated by the fact that dusts in the different places differ in composition, size and quantity, and the effect of simultaneous changes in these can not be predicted. A great deal of experimental work must be done on this problem, which up to the present time has received scant attention because of other work. A few isolated results indicate what may be expected from a careful study. It was found that an explosion could propagate through a strip of pure coal-dust 5 feet wide placed on the floor when the side-shelves were loaded with four times as much pure rock-dust. Where such a strip of coal-dust exists, because of spillage on haulageways, it must receive separate treatment except possibly where large quantities of rock-dust are present on overhead cross-timbers.

In another set of experiments it was found that a mixture containing 40 per cent of shale placed only on the floor would not propagate an explosion, nor would a mixture containing 65 per cent shale propagate an explosion when placed on cross and side shelves only. But propagation was obtained easily when both mixtures were placed simultaneously. To prevent propagation it was necessary to increase the shale content of the floor dust 5 per cent or that of the shelf dust 15 per cent. The only outstanding point in all the fragmentary work is that in no case was it necessary to have as high a percentage of rock-dust on the floor as on the ribs and roof. If fully substantiated this will be of great practical value, because in a given time the floor of a mine entry always receives the greatest contamination from coal-dust.

6. Source of ignition.- The term "source of ignition" is used to designate any arrangement of gas, explosives, or coal-dust either singly or in combination by means of which an explosion is projected into a dust mixture to determine the ability of that mixture to extinguish the explosion. While an unlimited number of sources can be devised under this definition, good judgment limits them to such as may be found in commercial mines. It has been found that the explosibility of coal-dust varies with the strength of the source of ignition. For example: the amount of rock dust required to prevent propagation of flame directly from a blow-out shot of 4 pounds of black blasting powder has been determined for a number of coals. This charge was selected as one not likely to be exceeded where coal is undercut. When 100 pounds of pure coal-dust is added in front of this blow-out shot, the arrangement becomes a standard propagation test, the igniting source is much stronger, and the amount of rock-dust required to prevent propagation is considerably increased. If a 50-foot zone of a 10 per cent natural gas-air mixture

ignited at the closed end is substituted for the cannon shot and coal-dust, a still stronger source is obtained and the amount of rock-dust required is increased again. There is, of course, a limit to this process at some point, but where this limit lies can not be said. A study of sources of ignition would require determination of (a) their dust-raising power and (b) their igniting power. Here again is a problem which has received scant attention, yet our knowledge of the explosibility of coal-dust is not complete until it is solved.

Practical applications will be possible only as the sources commonly found in mines are known. Fifteen years ago black blasting powder was responsible for many mine explosions. Blasting methods have been improved since that time and today electric arcs are the initiators of a majority of the mine explosions in this country. More commonly the arc or spark ignites a gas accumulation, and the gas explosion raises and ignites dust. There are, however, an alarming number of explosions from the ignition of dust clouds by electric arcs as an immediate sequel of haulage accidents which have occurred on pure intake air and in one or two cases close to the mine mouth. Quantitative studies of the initiation of dust explosions by gas explosions and by electric arcs have not yet been made in the experimental mine. A study of gas explosions opens a wide field of experimentation, because accumulations in mines are more likely to be nonuniform or stratified owing to the manner in which and the conditions under which the gas is liberated. Again, even with a uniform mixture the nature of a gas explosion will vary with the point at which the body is ignited and the degree of confinement to which it is subjected. It is evident, however, that more precise data on the initiation of coal-dust explosions must await a better knowledge of gas explosions in mines; it is also evident that thorough ventilation of gassy mines removes one of the principal means by which coal-dust explosions are started.

7. Surrounding conditions.- The factor of surrounding conditions includes all the physical influences, not a part of preceding factors, under which an explosion occurs. These conditions may be divided into (a) openings from or enlargements of the passageway in which the explosion is traveling, and (b) constrictions of or obstruction in this passageway. The former allow release of pressure and the latter cause retention of pressure. Intersection or branching of passageways belongs in the first, and bends in passageways belong in the second division. The effect of release of pressure ahead of an explosion is to cause an acceleration of the flame speed as it approaches the point of release. Release of pressure behind an explosion greatly retards the progress of the flame and in extreme cases may extinguish it. These phenomena have been demonstrated both in the experimental mine and in the gallery at Eskmeals, England, where a series of experiments were conducted on them. The results so far have been qualitative only.

EFFECT OF WATER ON EXPLOSIBILITY OF COAL-DUST

The foregoing items summarize briefly our knowledge of the explosibility of dry dust mixtures. Wet dusts can not be classified with dry dusts because of the tendency of the particles to cling together, making the formation of a dust cloud more difficult. Experiments have shown that the effectiveness of water in preventing scattering of dust is the only thing that can justify its use; at the working face there is certainly ample place for it. On the other hand, the use of water to render coal-dust nonexplosive in mine passageways has been a flat failure, and there is experimental evidence to show that it can not succeed with bituminous coals. Tests were made in 1925 of mixtures of four Utah coal-dusts and water. It was found in the course of this investigation that 20 per cent water was as much as these dusts (size 20 per cent through 200 mesh) would retain; if more was added it sank through the dust and drained away, confirming in general earlier similar tests on Pittsburgh bed coal-dust. Mixtures of two of the coals containing 20 per cent water propagated explosions under standard propagation test conditions. A similar mixture of the third coal was subjected to a gas explosion and also propagated flame throughout the test zone. These tests leave little doubt concerning the explosibility of wet bituminous coal-dusts. It is true that such wet dusts are raised into the air from the floor only with considerable difficulty, but they are dislodged from higher surfaces or timbers with ease, and when formed in a cloud by any means, they ignite and propagate flame despite the presence of the water.

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Picher Mining Co.

Circular No. 6113. April, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

METHOD AND COST OF MINING ZINC AND LEAD AT NO. 1 MINE, TRI-STATE ZINC AND LEAD DISTRICT, PICHER, OKLAHOMA.

BY WM. F. NETZEBAND

Washington, D. C.,
April, 1929.

This paper is the second of a series of publications dealing with mining methods and costs in the metal mines of the United States. The first paper, which discussed the Method and Costs of Mining Magnetite in the Mineville District, New York, was published as Information Circular 6092, and, like the present paper, dealt with an operation employing open stopes with pillar support.

Reports on 18 additional mines using the open stope method are in the course of preparation. After these reports are completed and published separately in Information Circular form, it is proposed to issue a bulletin in which will be discussed the open stope method with its variations, its application and limitations, and the costs of mining under the various conditions where it is employed.

Reports on other methods of mining are also being prepared, and it is planned to deal with these methods in a similar manner. These papers are all being written by officials and engineers of mining companies in accordance with an outline prepared by the Bureau of Mines for the purpose of obtaining uniform and comparable data.

Scott Turner

SCOTT TURNER,
Director.

Circular No. 6113,
April, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

METHOD AND COST OF MINING ZINC AND LEAD AT NO. 1 MINE, TRI-STATE ZINC AND LEAD DISTRICT [1]

By W. F. Netzeband[2]

INTRODUCTION

The purpose of this paper is to present to the operators in other districts a detailed description of the mining method used and the results obtained in exploiting one of the zinc-lead deposits of the Kansas-Oklahoma-Missouri district. No. 1 mine is located in the heart of the town of Picher in the extreme northeast corner of Oklahoma.

HISTORY

The first prospect drilling was done at No. 1 mine late in 1914, and shaft sinking was started early in 1915. Sufficient ore was developed to start construction of the mill in April, 1916. Production began in August, 1916, and the mine has been producing, except for short periods, to the present time. The property was originally a 40-acre tract, but additional properties have been consolidated with the original one until the mine now covers 200 acres.

The mill has a capacity of 1,400 tons in 24 hours and is one of the largest mills in the district. This mill was one of the first in the district to use flotation, a flotation plant having been added to the mill in the fall of 1917.

GEOLOGY

The surface rocks are of Pennsylvanian age, and are made up entirely of shales and sandstones of the Cherokee formation. The thickness of the Cherokee in the vicinity of No. 1 mine ranges from 60 to 100 feet.

The Cherokee formation lies unconformably upon the limestones and sandstones of the Chester formation, the youngest member of Mississippian age. This formation, which is entirely absent in many places, has a maximum thickness of 40 feet. In churn drill cuttings it is difficult to distinguish between the limestones of this formation and the underlying Boone formation, upon which it rests with a marked unconformity.

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1 The Bureau of Mines will welcome reprinting of this article but requests that the following footnote acknowledgment be made: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

2 One of the consulting engineers, U. S. Bureau of Mines.

The Boone formation is correlated with the Warwaw-Keokuk-Burlington series of other areas. It is the principal ore-bearing formation and consists of interbedded limestones and cherts. It is broken up locally into a number of members, the principal ones being the Short Creek oolite member, a thin but persistent oolitic limestone, and the Grand Falls chert member, a massive chert occurring 70 to 100 feet below the base of the Short Creek oolite. The thickness of the Boone ranges from 275 to 325 feet.

ORE DEPOSITS

The ore deposits of No. 1 mine occur in the brecciated and boulder ground 35 feet below the base of the Short Creek oolite.

This horizon is at the 270-foot or main mine level. There are two minor ore horizons mined, one at 230 feet, the Short Creek oolite horizon, and the other at 200 feet. The upper or 200-foot level is of the semisheet ground type; the ore occurs along the bedding planes and is disseminated throughout the blue and gray chert. The ore on the middle, or 230-foot level, is of the brecciated type.

The ore on the main level occurs disseminated in the jasperoid breccia, as massive lenses or patches and as the lining of vugs and cavities. The gangue minerals are jasperiod breccia, chert, calcite, and dolomite. The accompanying sketches (figs. 1 and 2) show the mode of occurrence of the ore in the different types of ground.

EXPLORATION AND ESTIMATION OF ORE RESERVES

There are no surface indications of ore in the district. All exploration work is done with the churn drill and is usually started by drilling several rows of holes across the tract from north to south or east to west at intervals of 200 to 400 feet, until a favorable area has been defined and then drilling the holes closer together.

The earlier holes were drilled 5-7/8 inches in diameter, but later the standard size was changed to 6$\frac{1}{4}$ inches. Logs of all holes are kept, and assays are made of all values above 2 per cent blende or 1 per cent galena; values below this are recorded as "shines." On the 200 acres constituting the No. 1 mine property 128,204 feet of drilling has been done. The drilling is all done on contract at prices ranging from $1.00 to $1.25 a foot depending upon the character of the ground being drilled.

Estimates of ore reserves are based on a careful analysis of the churn drill records. Past experience has shown that an ore-body mills out about 10 per cent better than the estimate. In estimating tonnage, a factor of 12.5 cubic feet per ton is used for rock in place.

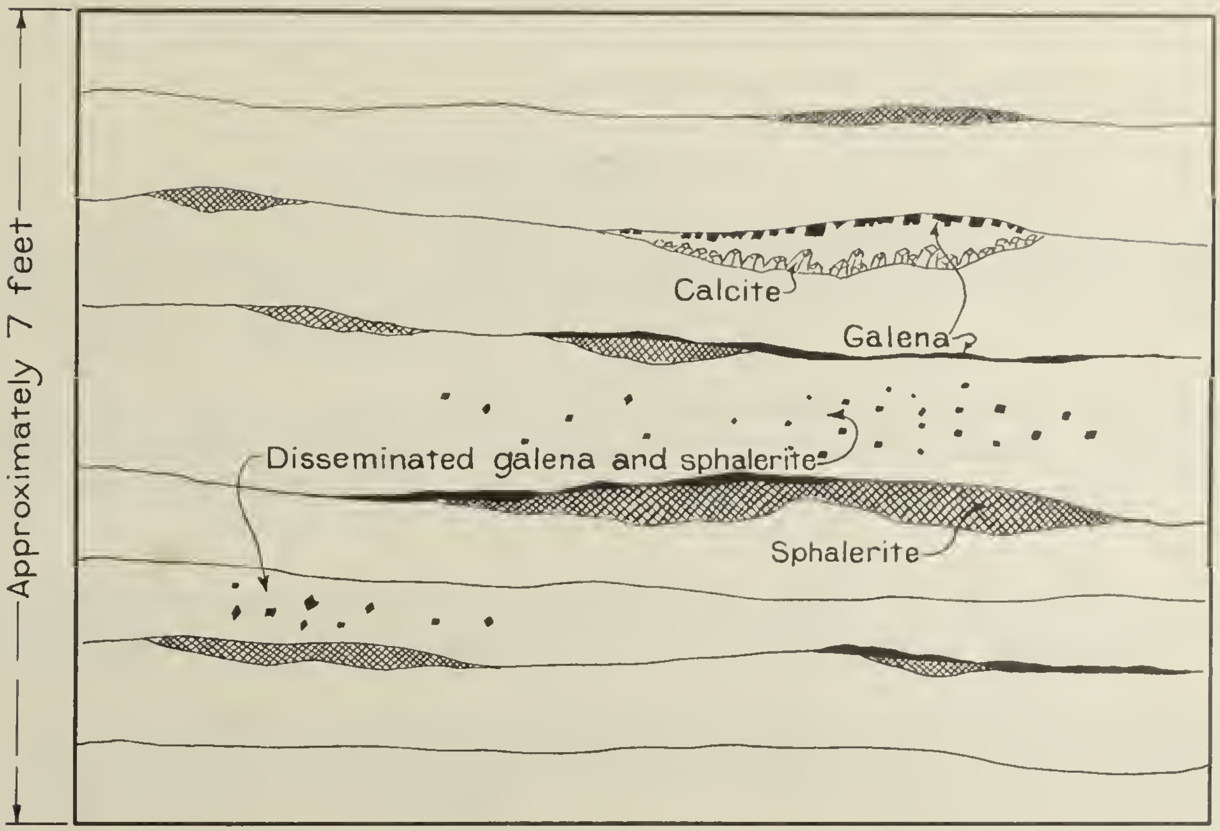

FIGURE 1.—Orebody of semi-sheet ground type

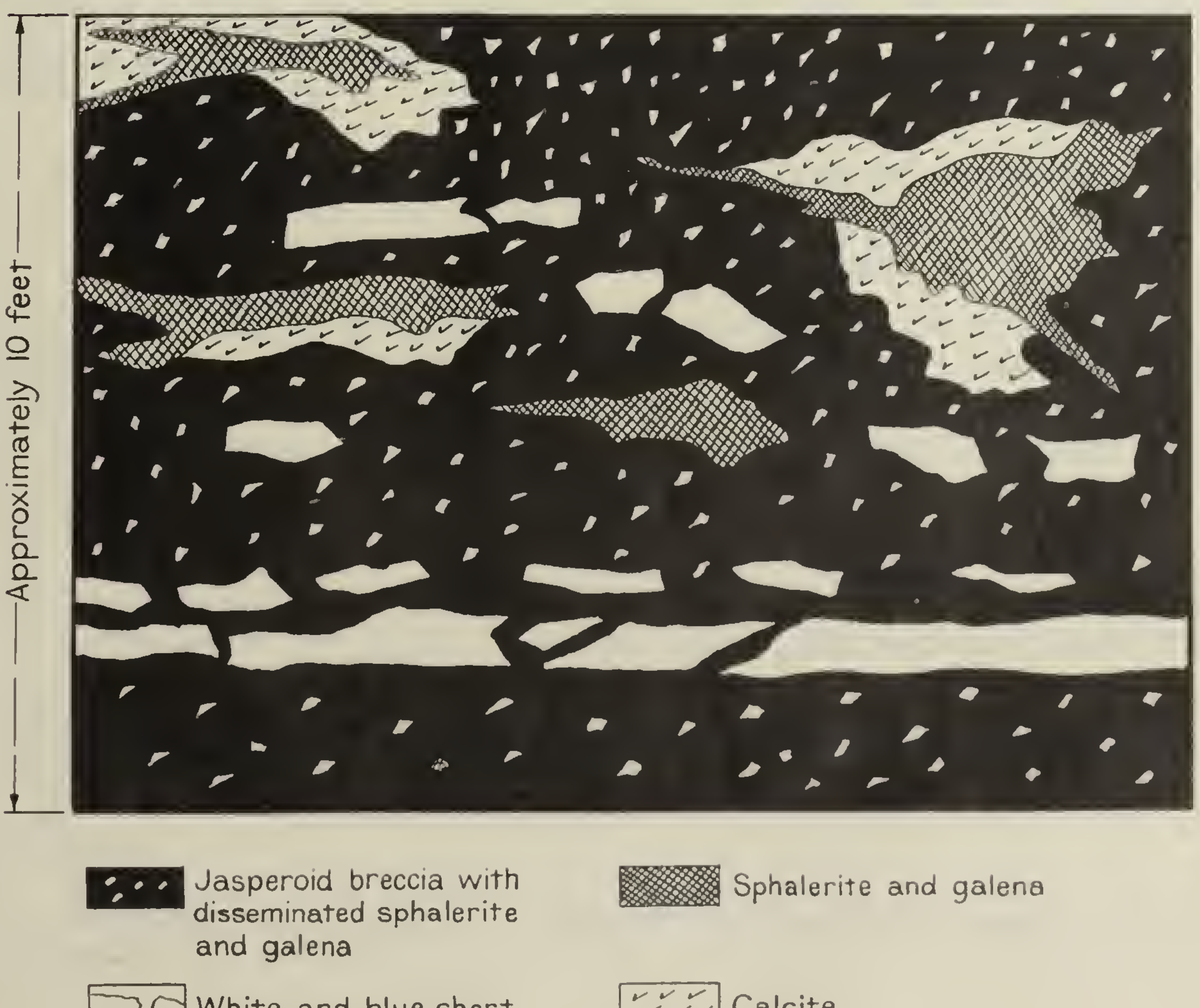

FIGURE 2.—Orebody of the brecciated type

EARLY MINING METHODS

The early mining methods were similar to the methods typical of the district; hand-shoveling into cans set on trucks, tramming by mule to the shaft, and hoisting the can of ore to the surface.

When the original No. 1 mine was consolidated with the first of the other properties, mechanical haulage with cars instead of cans was installed. A skip hoist was installed at the mill shaft and all ore hauled thereto by trolley locomotive.

Mechanical loaders of various types have been tried, but none have proved successful. Hand-loading has been found to be the most economical method.

As to the mining method, the open stope system with pillar support has continued to be used, since no other method has been found that could be employed satisfactorily in this type of ore deposit.

DEVELOPMENT SYSTEM

No definite development system is carried out once the shaft has been sunk to the ore and the mine opened up to the production stage. The churn drilling has defined the mineable areas, and the problem has resolved itself into keeping the pillars in the leaner ore as much as possible and at the same time providing adequate support to the roof.

Prospect drifts, known locally as "pull" drifts, are driven ahead of the workings as the occasion requires, but these are usually driven to an isolated, previously proved orebody, with the idea of later using them as haulage drifts.

Main Shaft.- The main or mill shaft was originally sunk at the mill site on the original tract. The shaft is 5 by 7 feet in cross section, the standard size for shafts in the district. It is sunk to the 270-foot level which is the main haulage level for the mine. Originally it was equipped with the usual type of Joplin hoist, using cans to bring the ore to the surface. The standard can holds about 1,400 pounds of ore.

The shaft is close-cribbed with 2 by 6 inch pine timbers from the collar of the shaft to a point below the shale. Careful lagging and packing behind the cribbing is necessary, since the shale slacks off and gives trouble when wet unless it is held securely in place so that it can not slump. Below the shale no cribbing is necessary unless loose boulder ground is encountered, in which case "jump" cribbing is tied in above and below to 6 by 6 inch bearing timbers set in hitches in solid rock.

The shaft was equipped for skips in 1920 using 2½-ton self-dumping skips and hoisting in balance.

Auxiliary Shafts.- The usual practice in the district is to sink a shaft to an isolated orebody and tram the ore on the surface, rather than to connect underground and hoist at a central shaft. For this reason the mines usually have two or more "field" shafts for hoisting ore besides the mill shaft. At No. 1 mine, however, all ore is hoisted at the mill shaft. Before the various properties were consolidated, several shafts were sunk on each tract, so that at present there are 14 "field" shafts on the 200 acres. The only one of these in use at present is equipped with a double-deck cage for handling men and material.

The early shafts were sunk by the company as a regular part of the mining operation, and due to the heavy flow of water the costs were excessive. The later shafts were sunk on contract, but as all of these were sunk a number of years ago, no costs are available. An average contract price is $12 per foot in shale and $18 per foot in the rock, the company furnishing everything but labor and powder.

Prospect or "Pull" Drifts.- Prospect or "pull" drifts are driven 7 by 7 feet in cross section. The work is almost always done on contract, the contractor furnishing all the labor and the company all the equipment. Occasionally a contract is let wherein the contractor furnishes everything but the track and piping.

The average round with the amounts of powder used is shown in the sketch (fig. 3). The rounds are varied somewhat to meet the changing conditions of the ground. When the ground becomes harder, three or four cut holes are used, and occasionally it is found necessary to drill two or three extra bottom or "stope" holes in order to assure an even grade.

The powder used in $1\frac{1}{4}$ by 8 inches, 33 per cent, gelatin dynamite. The primers are made up with No. 6 caps; the caps are placed in the center of the stick of powder and the entire primer is put into safety tubes provided for the purpose.

The holes are drilled from a column set in the center of the drift; a heavy Leyner-type drill is used. All Leyner drills are operated by two men, a runner and a helper. The cut holes are drilled with 7-foot steels to a depth of 6 feet, and all other holes are drilled with 9-foot steels to a depth of 8 feet. Usually 6 feet are broken per round.

The costs given below are based on a drift which was contracted for at the rate of $7.50 per foot, with the contractor furnishing only the labor and the company furnishing everything else.

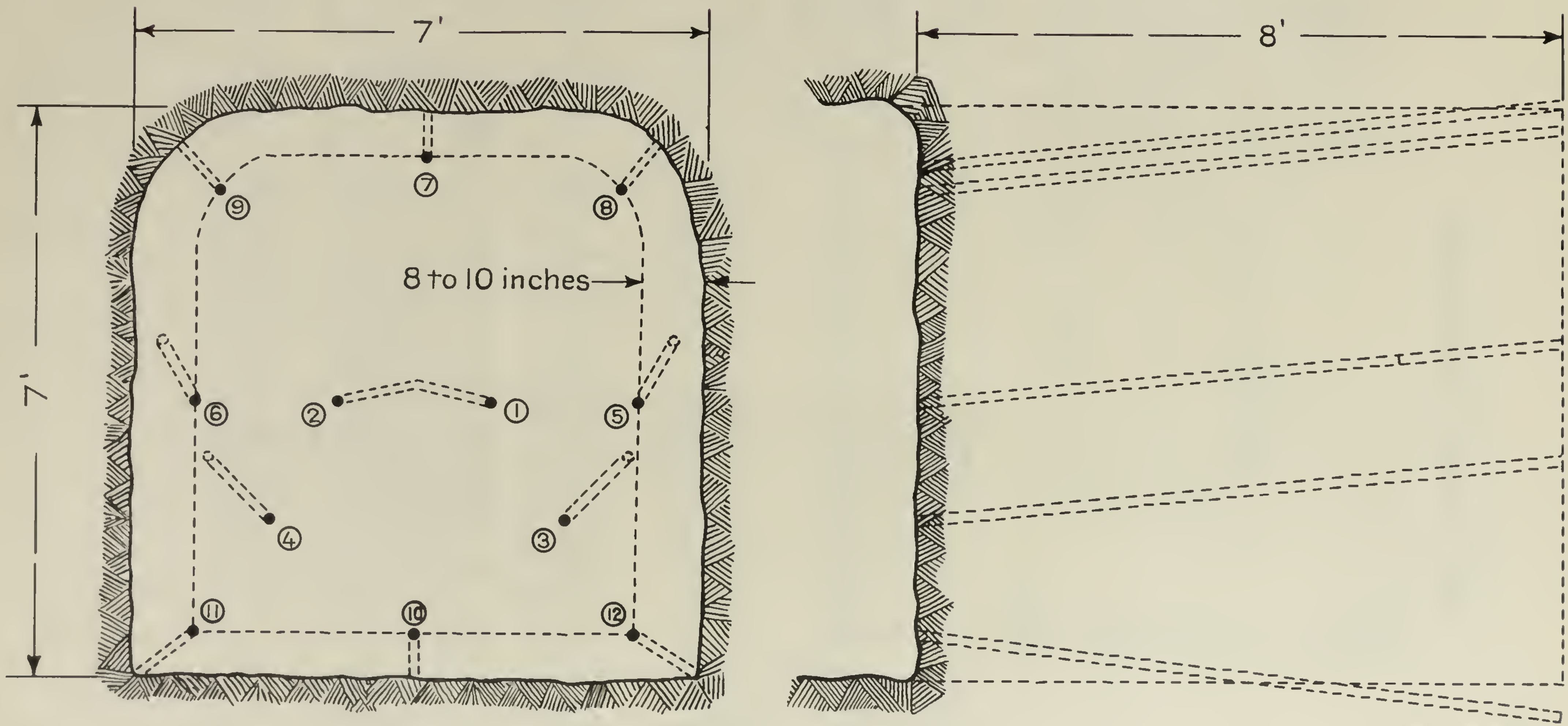

FIGURE 3.—Plan and profile of "pull drift" round

Order of firing "pull drift" round			
First	1 2	6 ft. deep	20 to 30 sticks 35 % gelatin
Second	3 4	8 " "	15 " 20 " " "
Third	5 6	8 " "	10 " " "
Fourth	7 then 8 9	8 " "	10 " " "
Fifth	10 then 11 12	8 " "	12 to 15 " " "

Cost per Foot of Prospect Drift

	Cost	Per cent
Drilling labor	$2.95	20.8
Mucking labor	4.42	31.3
Explosives	2.38	16.8
Tramming (power)	.97	6.9
Labor insurance carried by contractor	.48	3.4
Compressor	.62	4.4
Tracking and piping	.46	3.3
Ventilation	.43	3.0
Steel consumption	.06	0.4
Blacksmith	.70	5.0
Miscellaneous (carbide, repairs, etc.)	.66	4.7
Total cost per foot	$14.13	100.0

When a new orebody is being developed and the rock has to be hoisted to the surface, there is an additional cost of 14 cents per foot for hoisting.

Raises.- Raises are not necessary very often, but when an isolated orebody is found above the main level a raise is driven up, usually at an angle of about 45° so that it can later be used for an ore chute. The raises are driven 6 by 6 feet in cross section.

The raises are drilled from a shaft bar with the same type of heavy Leyner drill that is used in the "pull" drifts. The round used is practically the same as for the "pull" drift, except that the center stope and roof holes are not used. This cuts down the powder consumption slightly.

The contract price is $3.50 per foot, but the contractor furnishes only the drilling labor. The company does the mucking and tramming and supplies all equipment. These costs will be practically the same for raises as for "pull" drifts.

After the upper orebody has been developed to assure mining on that level, the raise is converted into an ore chute by putting a front on the lower end with a gate for loading. If the raise is the only means of entrance to the new ore level, a partition is built so that part can be used for a manway. Usually, however, two raises are driven, one to be used for a manway. This arrangement always affords better ventilation, and often a raise is driven solely for this purpose.

Formerly all raises were driven vertically with a light stoper, but the inclined raise has been found so satisfactory that no vertical raises are now driven.

PRESENT MINING METHOD

The plan and sections of No. 1 mine are shown in the sketches (figs. 4 and 5). The headings are kept well in advance of the main stope except where the ore face is low, when no stope is carried, and the entire face is advanced by the same method as the heading.

The chert and jasperoid breccia breaks into large boulders, and it is necessary to do a considerable amount of "boulder popping" before loading. All boulders are drilled with a light jackhammer before blasting.

For all drilling except the "boulder popping," the heavy Leyner drill is used. The headings are drilled from a post, but the stopes are drilled from a tripod. For ordinary work 33 per cent ammonia powder is used, but for wet work or where the ventilation is poor, gelatin powder of the same strength is used. The cost of ammonia powder averaged $0.1266 per pound and the gelatin powder $0.1400 per pound for the year 1928.

Underground Support.- The mine is supported almost entirely by pillars of ore from 20 to 60 feet in diameter, the average being 30 feet. The size depends upon the character of the ground and the height of roof. The pillars are spaced 40 to 100 feet, center to center, with an average spacing of 80 feet. Wherever the character of the ground permits, the roof is arched between pillars.

Some of the pillars have been trimmed down as small as was considered safe; these have shown no signs of taking weight. At present the pillars represent about 15 per cent of the total area cut, but many of these will be recovered before the mine is abandoned. All pillars can not be recovered, for the mine is under the town of Picher and the surface must be protected.

There are parts of the mine where thick shale or loose boulder ground necessitates timbering, but generally these parts are not ore-bearing and only "pull" drifts are driven through them. This method of support is of minor importance in the mining operations.

DRILLING AND BLASTING PRACTICE

Compressors.- Air is furnished from a central compressor plant serving several mines. The capacity of this plant is 9,380 cubic feet per minute. The individual compressors range in capacity from 1,080 to 4,700 cubic feet. The air is maintained at 110 pounds at the plant.

No. 1 mine consumes about 1,600 cubic feet of air per minute. The pressure at the machines ranges from 85 to 90 pounds, the average being about 86 pounds.

Drills.- The drills used are all of the heavy Leyner type, although several makes of machines are used. They are run from tripods or columns. All drills use $1\frac{1}{4}$-inch hollow-round steel. Jackhammers are employed for "boulder popping," using 1-inch hexagon, hollow steel.

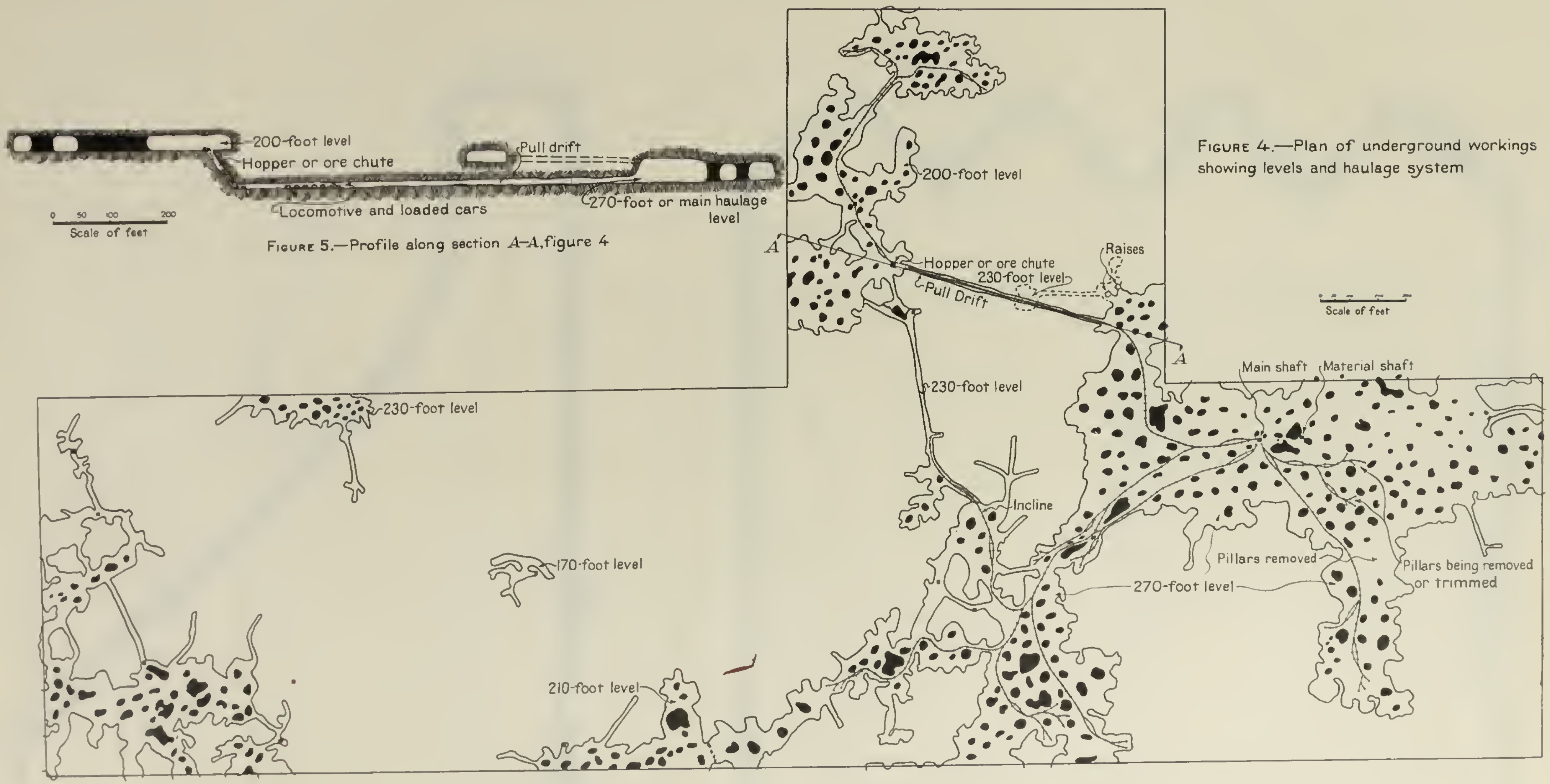

FIGURE 4.—Plan of underground workings showing levels and haulage system

FIGURE 5.—Profile along section *A–A*, figure 4

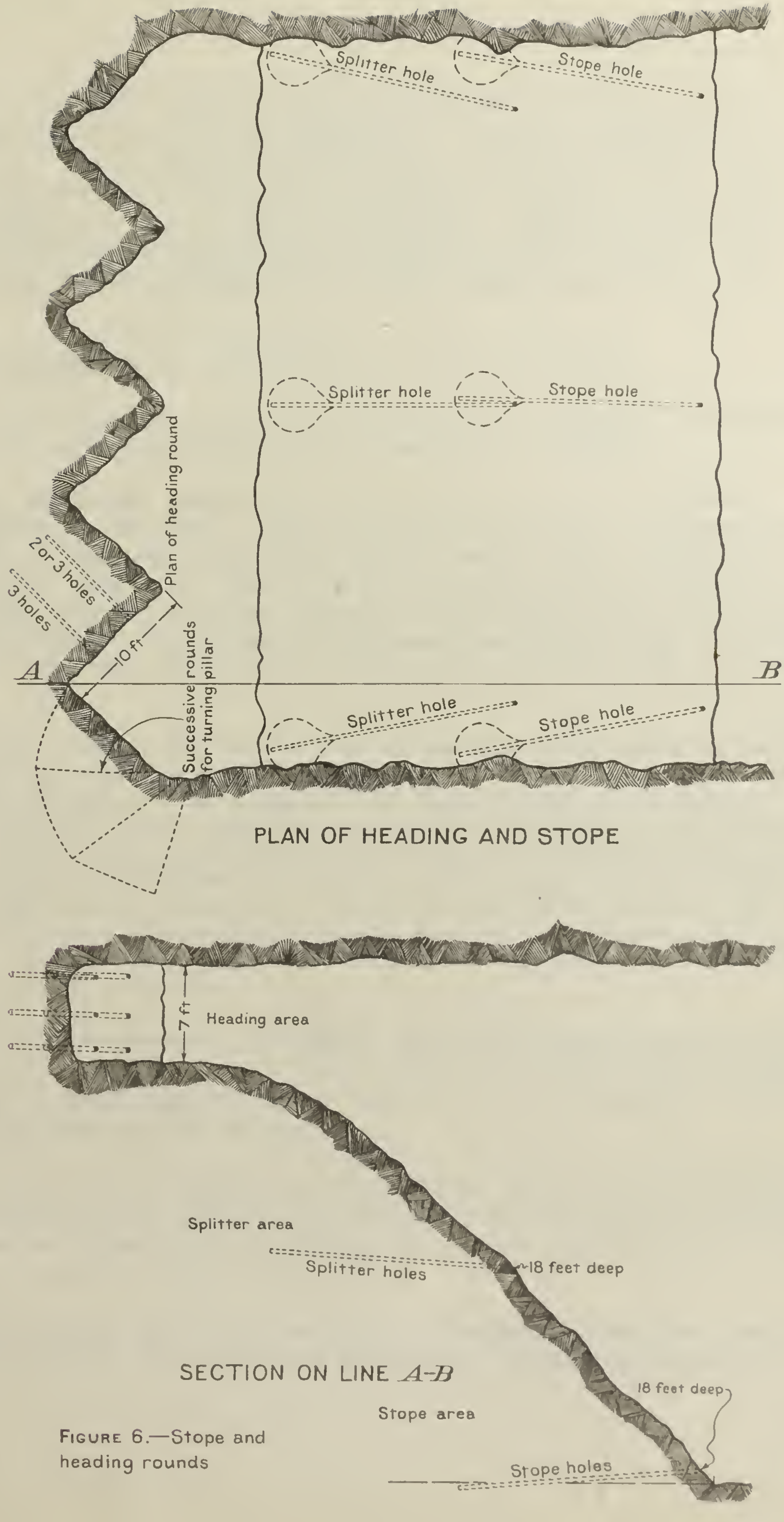

FIGURE 6.—Stope and heading rounds

The standard cross bit with 18° taper and 1/4-inch difference in gauge is used for both the Leyner drills and the jackhammers. The steel lengths for the Leyner drills are 5, 7, 9, 11, 13, 15, 17, and 20 feet. The holes are started with 2 3/4-inch gauge and the 20-foot holes bottomed with $1\frac{1}{2}$-inch gauge. The jackhammer steels are made up into 1, 3, and 5 foot lengths. The holes are started with 2-inch gauge and the 5-foot holes are bottomed with $1\frac{1}{2}$-inch gauge. The steels are bitted and shanked underground at a central plant located near the material shaft.

The steel consumption is 0.073 pounds per ton of ore mined.

Blasting.- The blasting practice in the district is different from that at most metal mines; so the practice will be described in some detail.

The heading rounds are loaded with three-fourths of a box of ammonia powder, 12 to 14 sticks to the hole. When the ground is hard and tight and a six-hole round is used, the holes are fired in the order shown in the diagram (fig. 6). In ordinary ground, No. 1 hole is omitted and No. 2 hole is placed vertically above No. 3. In this case No. 2 is the lead hole and the other holes are fired in the same sequence as before. Where the face is less than 12 feet high only a heading round is used. All holes are stemmed, using a clay cartridge prepared for the purpose.

For mine faces ranging from 25 to 40 feet high the round shown in the diagram is used. For mine faces above 40 feet two sets of splitters are used, making 6 splitters and 3 stope holes. The slope of the stope is kept at an angle of about 45°. For the 40-foot face, which will be taken as a model in the following description, the collars of the splitter holes are about 20 feet horizontally behind the heading round, and likewise the collars of the stope holes are about 20 feet behind the collars of the splitters. Where the mine faces are lower the measurements are cut down accordingly. For faces over 40 feet high six splitter holes, in two rows of three each, and three stope holes are used. In every case the burden is the same on each hole.

The splitter holes are three in number, one in the center and one on either side, all in the same horizontal plane. These holes are drilled with a rise of 1 inch to the foot so that they can be readily washed of all cuttings after squibbing. The holes are drilled with 20-foot steels.

The stope holes are also three in number and are placed vertically under the splitters. These holes are drilled down with the same fall that the splitters have rise. They are so located that the bottom of the hole will be below the grade of the drift so as to prevent high bottom.

Each splitter and stope hole is squibbed or chambered three times before the final charge is loaded. The first time 15 sticks of powder are used. The

hole is then thoroughly washed and 24 hours later is shot with 30 sticks of powder. After again being thoroughly washed, the hole is loaded with from 45 sticks to 1 box of powder, depending on the size of the pocket. The hole is once more thoroughly washed, preparatory to loading the breaking charge. For this charge from 2 to 4 boxes of powder are used. The hole is very seldom loaded to the collar except where the ground at the collar is unusually tight because of soft or cavey ground between the collar and the pocket. All holes are stemmed, using a clay cartridge prepared from clay found on one of the company's properties. The cartridges cost about $0.01 each.

An average stope or splitter hole will break about 250 tons of ore. The powder consumption is 0.75 pounds per ton.

A heading round will break 40 tons per machine shift, and where stope and splitter holes are used, 60 tons per machine shift are broken. For high headings (70 ft.) 75 tons per machine shift has been broken.

LOADING AND TRAMMING

All loading is by hand labor. The cars are of 20-cubic-foot capacity and average 1 ton of ore. The size of the car is 4 by $2\frac{1}{2}$ by 2 feet. The track gauge is 24 inches throughout the mine.

The cars are gathered on "lay-bys" or sidings near the working face by trammers who help the shovelers with the loaded cars. One trammer or "mule" is provided for every four shovelers. For short hauls mules are used to bring the cars to the shaft or hopper, but for long hauls electric locomotives of the trolle type are used. Three mules are used underground and two locomotives, each of whic weighs $4\frac{1}{2}$ tons and is capable of handling 40 loaded cars.

Trammers or "bumpers" take the cars from the "lay-bys" near the skip pocke (fig. 7) and dump the ore upon a grizzly made of 90-pound rails set 6 inches apar above the skip pocket. "Screen apes" sledge all boulders which will not pass through the grizzly.

All ore mined on the 200-foot level in the north part of the mine is dumpe into an ore chute or hopper and drawn out on the main haulage level. This is the only hopper in use at present. The hopper is a raise driven at an angle of 45° with the lower end equipped with gates for drawing off the ore. The gates are raised and lowered by a simple lever arrangement.

PUMPING

The only pumping necessary is from a sump which drains the skip pocket. The water level has been lowered for the district so that the main level is dry. To keep the sump dry, two small pumps with a combined capacity of 100 gallons per minute are run for about three hours each day.

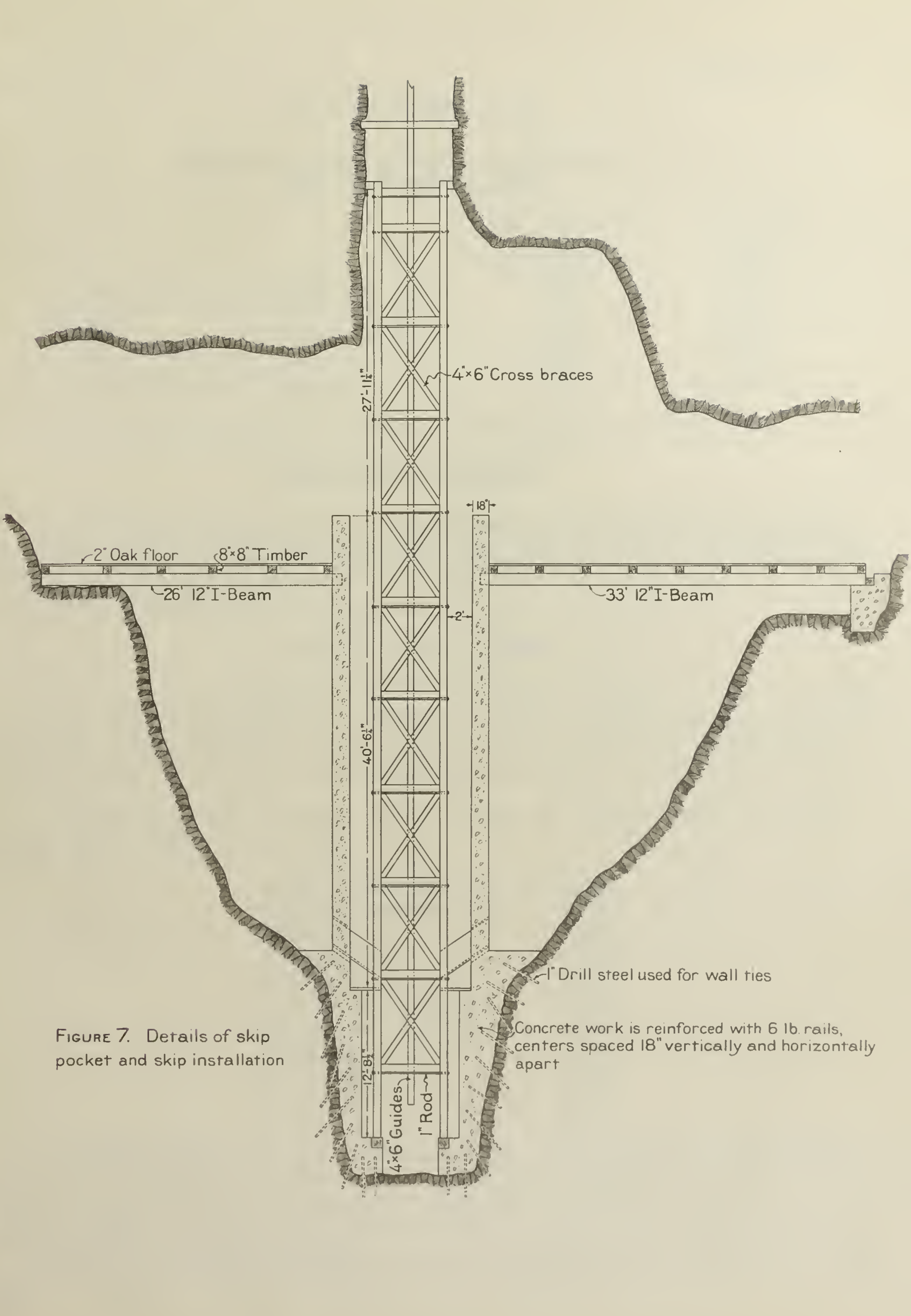

FIGURE 7. Details of skip pocket and skip installation

LABOR EFFICIENCY, 1928

Record of Labor Performed at Mine No. 1, July to December, 1928

	Total shifts for 6 months	Tons per Shift
Trammers	2,125	32.36
Drill runners and helpers	3,000	22.92
Muckers	3,500	19.65
Miscellaneous	3,000	22.92
Total underground operations	11,625	5.91

Ore hoisted during last 6 months of 1928 amounted to 68,770 tons.

PERCENTAGE OF EXTRACTION

At present 15 per cent of the area mined is left in pillars, but it is expected that after the mine is abandoned 90 per cent of the ore, at least, will have been extracted and that most of the remaining 10 per cent will be in lean ore and will not represent a very great loss.

WAGE AND CONTRACT SYSTEM

All labor except mucking is based on an eight-hour day. Muckers are paid on contract, $14\frac{1}{2}$ cents per car if they work less than 6 shifts per week, and $15\frac{1}{2}$ cents per car if they work the full 6 shifts. When loading from a large pile, with little clean-up work, the average mucker will load 30 to 40 tons per shift.

The following wage schedule was in effect for 1928 when zinc ore prices stayed at $40 or under for the year.

Machine runners	$4.25
Machine helpers	3.75
Trammers	3.50
Blacksmiths	4.25
Hoistmen	4.75
Locomotive operators	4.25
Locomotive brakemen	3.75
Powdermen	4.50
Roof trimmers	4.00
Screenmen	3.50

This wage scale is based on $40 zinc ore. If the price goes above $45 and stays there for one week, all wages are automatically raised 25 cents per shift and muckers are raised 1/2 cent on the car. Likewise for every $5 raise in the price of ore above $45, the wages are raised at the same rate, but for every drop of $5 the wages are reduced accordingly. A week is always allowed between the wage changes to make sure that the market will not fluctuate above or below the critical price.

VENTILATION

The ventilation is natural except in rare cases where a long drift is being driven or an orebody opened up from a raise. In such cases ventilation is supplied by small blowers driven by motors. Where the drift or orebody cuts a churn drill hole, this hole is cleaned out and the blower installed on the surface. If no drill hole is available, canvas tubing is used and the blower set out in the main workings.

FIRE HAZARDS

So little timber is used underground that the fire hazard is practically nil. In case of fire at the surface buildings, the workings are cut into other properties so that the men are never in danger.

SAFETY METHODS

Much attention has been given to safety during the past few years. The company maintains a full-time safety engineer and assistant. These men have full power to enforce any safety measures they deem necessary.

The Tri-State Zinc and Lead Producers Association also maintains a full-time safety engineer to cooperate with and coordinate the work of the various company men. A system of presenting flags to the various mines for not losing any time from accidents for periods of 3, 6, and 12 months has been inaugurated by the association. This has aroused the men's interest in safety measures, as is shown by the fact that mines from this district won the Explosive Engineer's Safety Trophy for the years 1926 and 1927 and that several mines from the district were in the runner-up list for 1928.

The company further arouses the interest of the men by paying all ground bosses a bonus of $25 for every month passed without a lost-time accident, and the men are given a pair of cotton work gloves.

MINING COSTS

Below is the total cost for delivering a ton of ore to the surface at No. 1 mine:

Cost of delivering 1 ton of ore to the surface at No. 1 mine

	Cost	Per cent
Total underground labor	$0.598	60.7
Supervision	.026	2.6
Compressors, air drills, drill steel	.127	12.9
Electric power	.041	4.2
Explosives	.110	11.2
Other supplies	.083	8.4
Total Cost	$0.985	100.0

Summary of costs in units of labor, power, and supplies, July to Dec., 1928

Name or number of mine ------ No. 1 Year, 1928 - 6 months' operation

Tons of ore mined and hoisted: 68,770

Mining method: Open stopes with pillar support

(A) Labor (man hours per ton):

	Cost	Per cent
Breaking (drilling and blasting)	0.377	28.1
Mucking	0.407	30.4
Haulage and hoisting	0.282	21.0
Supervision	0.028	2.1
General	0.247	18.4
Total labor underground	1.341	100.0

Average tons per man per shift	5.91
Labor, percentage of total cost	63.35

(B) Power and supplies:

Explosives (lbs. per ton) - 33% ammonia and gelatin		0.750
Total power (hp. hrs. per ton)		9.91
(1) Air compression	6.00	
(2) Hoisting	2.77	
(3) Pumping	0.22	
(4) Haulage (includes lighting)	0.92	
Other supplies in percentage of total supplies and power		22.99
Supplies and power, percentage of total cost		36.65

* * * * * *

I. C. 6119 May, 1929

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

METHOD AND COST OF MINING THE THICK FREEPORT COAL IN A WESTERN PENNSYLVANIA MINE

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METHOD AND COST OF MINING THE THICK FREEPORT COAL IN A WESTERN PENNSYLVANIA MINE [1]

By J. W. Paul[2] and H. Tomlinson[3]

This report is one of a series of papers on coal mining methods and costs which are being prepared through the sponsorship of the U. S. Bureau of Mines for the purpose of bringing to the industry a knowledge of methods as they are used at the present time, and data on certain features which govern costs, safety, and conservation.

The methods employed are influenced by local conditions with respect to the character and dip of the coal, the nature of the roof and the thickness of the overburden, and the means employed for roof support and its control.

Mining costs are governed by the relative efficiency in applying the methods employed underground as well as by the selection of the method best adaptable to the physical conditions encountered.

This paper presents a detailed description of the methods used and the results obtained in the operation of a mine in the Thick Freeport coal bed in western Pennsylvania. classed as a captive mine; the product of this mine is consumed by the parent company which is engaged in furnishing electric power, commonly known as a public service utility.

ACKNOWLEDGMENTS

Through the courtesy of the general superintendent and the cooperation of the operating staff of the mine in question, the details given herein have been made possible. This paper is printed with the approval of the executive official, who has checked the accuracy of the methods and data given.

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1 - The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be made: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"
2 - Senior mining engineer, U. S. Bureau of Mines.
3 - Associate mining engineer, U. S. Bureau of Mines.

HISTORY

This mine was opened in 1920, since which date 3,333,000 tons of coal have been produced up to and including 1928. The Thick Freeport coal in the district in which this mine is located has been developed by other mines on properties adjoining, one of the earliest of which was opened in 1901. Therefore, the quality and character of the coal had been established prior to the development of the mine under discussion.

GEOLOGY AND TOPOGRAPHY

The rocks of Allegheny County are embraced in the Washington, Monongahela, Conemaugh, and Allegheny groups of the Carboniferous Age. The Allegheny, of which the Thick Freeport coal bed is the top member, is the lowest of these groups and has a limited outcrop on the Allegheny River north of Pittsburgh. The measures are composed of massive sandstones, shale, thin limestones, and several beds of coal and clay. Since these measures are below the Thick Freeport coal, they are of no special importance in relation to development of this coal. The Conemaugh group above the coal consists of grey, red, and greenish shales, sandstones, limestones, and a number of coal beds, some of which are workable in other districts.

The immediate member of this group over the coal is the Mahoning sandstone which is coarse grained and hard, ranging up to 40 feet thick; the breaking of this member is one of the problems in the withdrawal of pillars. A generalized section of the measures above the coal is shown in Figure 1. This, however, does not include all the measures, which have a total thickness of 750 feet and which include a part of the Monongahela group, extending 50 to 80 feet above the Pittsburgh coal.

Between the Mahoning sandstone and the coal is a series of shale, bone, and carbonaceous shale forming the immediate roof, and it is this material that requires timber support in the advance mining work. At intervals, but without any regularity, this immediate roof material disappears, and the sandstone is found resting on the coal; in places the sandstone replaces a part of the coal bed, which adds to the expense of development, and where the coal is too thin for economic mining, the leaving of areas of thin coal adds to the difficulty of bringing-on roof subsidence in the pillar work. The replacement of the coal by the sandstone, forming what is known as "want" areas, is a notable characteristic in mines developed in the Thick Freeport coal.

A detailed section of the coal, immediate roof, and floor is shown in Figure 2. Figure 3 shows a washout.

<u>Faults, dip</u>.- There are no true displacement faults in the coal, and the dip averages 1 per cent, with occasional pitches of 6 per cent for short distances and a return to the original gradient.

<u>Floor</u>.- The immediate floor under the coal bed is a fire clay which ranges from 4 to 10 feet in thickness and which heaves very slightly along the line of pillar extraction. In the solid work no heaving action has taken effect.

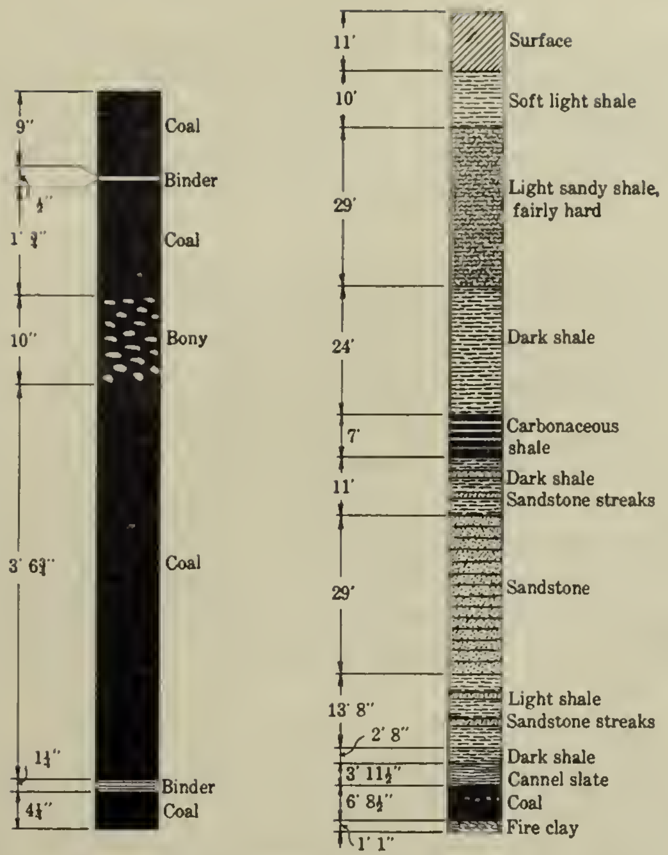

FIGURE 1. - LOG OF BOREHOLE AND SECTION OF THE COAL PED AS RECORDED FROM THE LOG

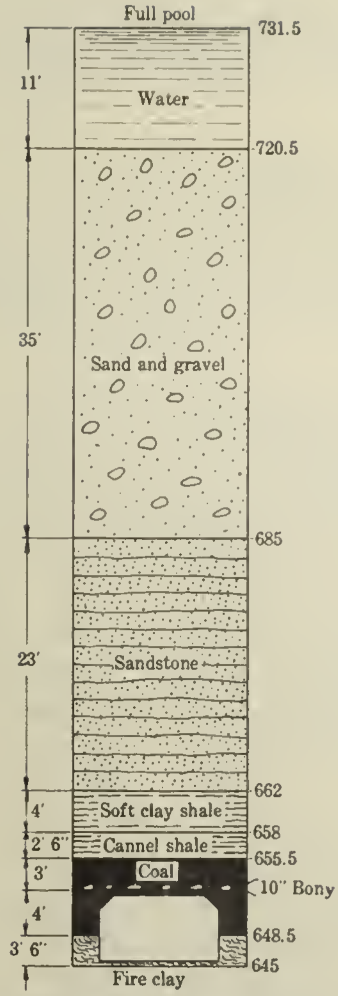

FIGURE 4. - TYPICAL CROSS SECTION OF ONE OF THE TUNNELS AND THE ROOF UNDER THE RIVER

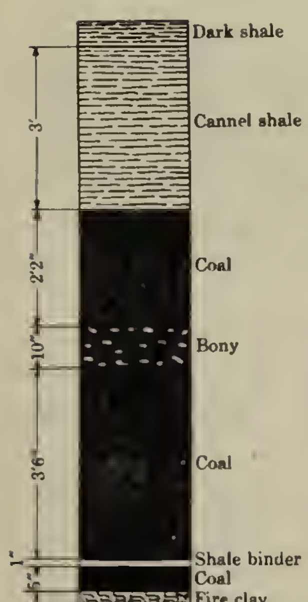

FIGURE 2. - AVERAGE SECTION OF THE COAL BED AND IMMEDIATE ROOF

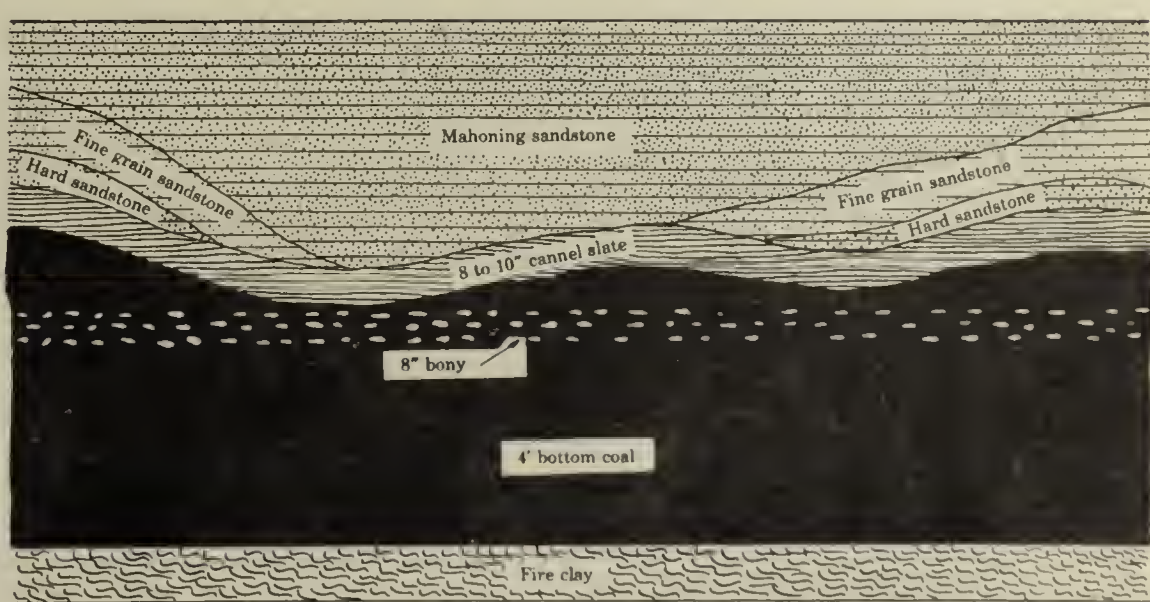

FIGURE 3. - A WASHOUT OF THE UPPER BENCH OF THE COAL, LOCALLY TERMED A "WANT"

Water.- Very little water is encountered in the solid or pillar workings, and such as is present has an alkali reaction. The greater quantity of water, amounting to 350 gallons per minute, comes from the vicinity of the shaft and from a gravel bed overlying the entries near the shaft.

In mines in the same coal bed within a radius of 5 miles, previously developed, there has not been any indication of gas pressure within the coal or adjacent strata, although in unventilated parts gas has accumulated; therefore, the ventilation of this mine presented no unusual problem, except to provide ample capacity and means for conducting the air under low pressure.

The coal bed.- The coal bed, which averages 7 feet in thickness, has three distinguishing benches - the lower, which is persistent and averages 4 feet, the bony which lies near the middle and is persistent, and the upper bench which is about 2 feet 2 inches but varies in thickness; where the thickness of the bed becomes less than normal it is always at the expense of the upper bench. In the "want" areas it is the upper bench that is lacking in its full or partial development. The character and quality of the coal is such that it may be used for by-product coking, steam, and for domestic and gas-making purposes. However, the entire product of the mine is used for steam generation. An average proximate analysis of the upper and lower benches is as follows: Moisture, 2.4; volatile matter, 34.5; fixed carbon, 55.0; ash, 8.1.

The coal has an inch shale band about 5 inches from the bottom and a bed of bony 8 to 10 inches thick about 4 feet from the bottom. Above the bony the coal averages about 2 feet 2 inches. Immediately above this is a cannel shale about 3 feet thick which forms the immediate roof. The bony near the middle of the bed is relatively high in combustible material and gives about 10,000 B.t.u., but it is discarded, although when pulverized and mixed with coal in similar form it would prove to be a source of power.

SHAFTS AND TUNNELS

Shafts.- Access to the mine is by means of a hoisting shaft 162 feet deep and by an air shaft and double deck slope which are 3,300 feet from the hoist shaft. The shaft has two compartments and is oval in cross section, 16 feet by 20 feet. The sandstone over the coal is overlain with glacial drift and sand; in sinking the shaft the Moran type of air lock was employed under 25 pounds' pressure down to the sandstone, but from this point the regular open cut method was employed to reach the coal which has an elevation of 642 feet A.T.

The slope is dipping 33-1/3 per cent, and the lower compartment is 12 feet wide by 12 feet high, 542 feet long; it is used as an intake airway and is provided with a track for the transport of materials. The upper compartment is 12 feet wide and 7 feet high and is used as a manway; it is equipped with a continuous concrete stairway and in it are placed the electric power lines well protected. This upper compartment connects at its bottom with the manway entry. The air shaft is approximately 300 feet south from the slope, 15 feet in diameter, 164 feet deep, and is concrete lined. The main hoisting shaft has one compartment in which tandem skips operate in balance; this shaft has also a second compartment equipped with a winding concrete stairway and having space for pipes at either side.

Tunnel construction.- The coal acreage under development is on the opposite side of a river from the hoisting shaft and preparation plant and lies 80 feet below the bottom of the river; to connect the mine with the hoisting shaft two parallel tunnels were driven in the coal under the river. The overlying sandstone is 28 feet thick and forms the main roof over the tunnels, as shown in Figures 4 and 5. These tunnels are 10-1/2 feet wide by 7 feet high and 1,400 feet long between shore lines of the river. Such water as was intercepted was controlled by grouting and some of the crevices required from 400 to 600 bags of cement before the water was shut off; 15 to 20 holes in the roof were required in some instances and an air pressure of 80 to 160 pounds was used. Sets of 10 by 10 inch square timber were installed in parts of the tunnels, the roof at most places being self-supporting.

The Hoist.- A balanced, tandem skip hoist is in service in the hoisting shaft. The headframe is of steel construction and the winding drum is operated by an electric motor. Each of the skips have two compartments, one above the other, and are loaded from single car capacity bins located below the two car rotary dump at the foot of the shaft, as illustrated in Figure 6.

The product of the mine when delivered to the top of the shaft is taken over by the parent company, and after being passed through a crushing plant is taken to the point of consumption nearby.

METHOD OF DEVELOPMENT AND MINING

(a) The general plan of development has been the regular entry room-and-pillar method laid out in panels, Figure 7. From the foot of the slope and air shaft seven parallel entries 11 feet wide and on 50-foot centers were driven in the coal. Four of these are used as air courses, two as haulages, and one as a manway. From these entries at intervals of 1,800 feet four face entries, 11 feet wide and on 50-foot centers, have been turned at right angles. Barrier pillars 200 feet wide have been left on either side of all main and face entries. From the face entries pairs of butt entries have been turned at intervals of 300 feet. These butt entries are driven to their limit, after which rooms 15 to 18 feet wide are driven from one entry on 90-foot centers. Figure 8 shows the plan of the development of five butt entries with the periodic break lines indicated extending across as many as five pairs of butt entries, and is an exhibition of successful panel-retreating method of mining.

When the rooms have advanced beyond the second crosscut a bisecting room is started in this crosscut which then places the rooms on 45-foot centers. The advancement of the rooms is so timed that the face of each lags behind the adjacent room by 52 feet; when the room has advanced its full distance, plus the distance through the chain pillar of the next butt entry, the face is then on the main break line which is on an angle of about 49° degrees with the butt entry, and the room pillar is ready to be brought back. See Figure 9. The break lines are about 1,800 feet long and extend across four pairs of butt entries.

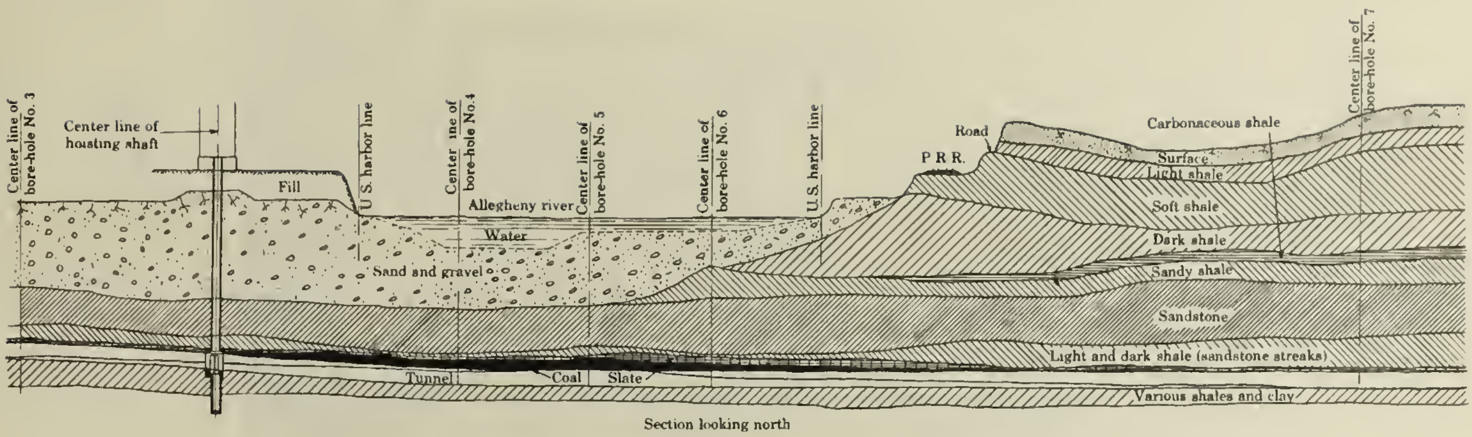

FIGURE 5. - PROFILE SHOWING STRATA ABOVE THE RIVER TUNNELS

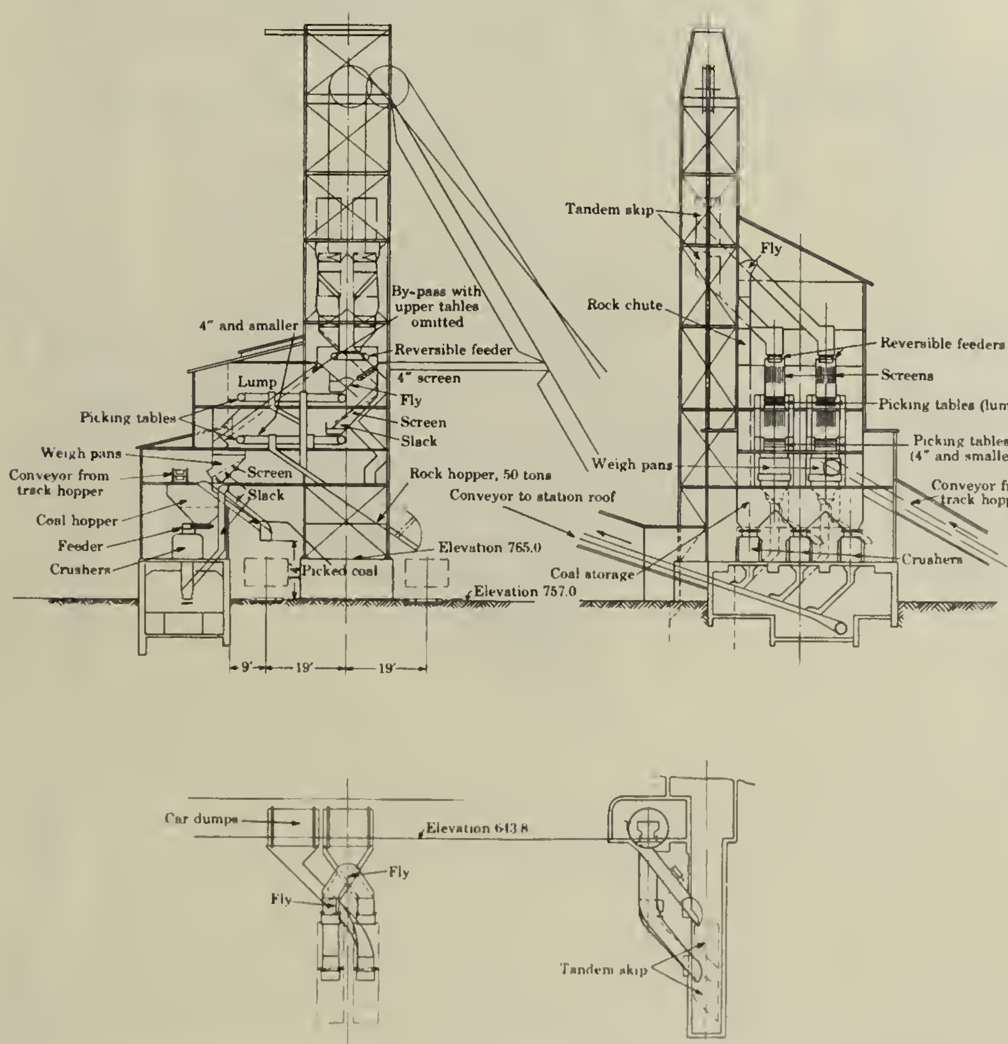

FIGURE 6 - HEADFRAME AND TIPPLE AT SURFACE AND DUMP AND SKIP LOADING AT SHAFT BOTTOM

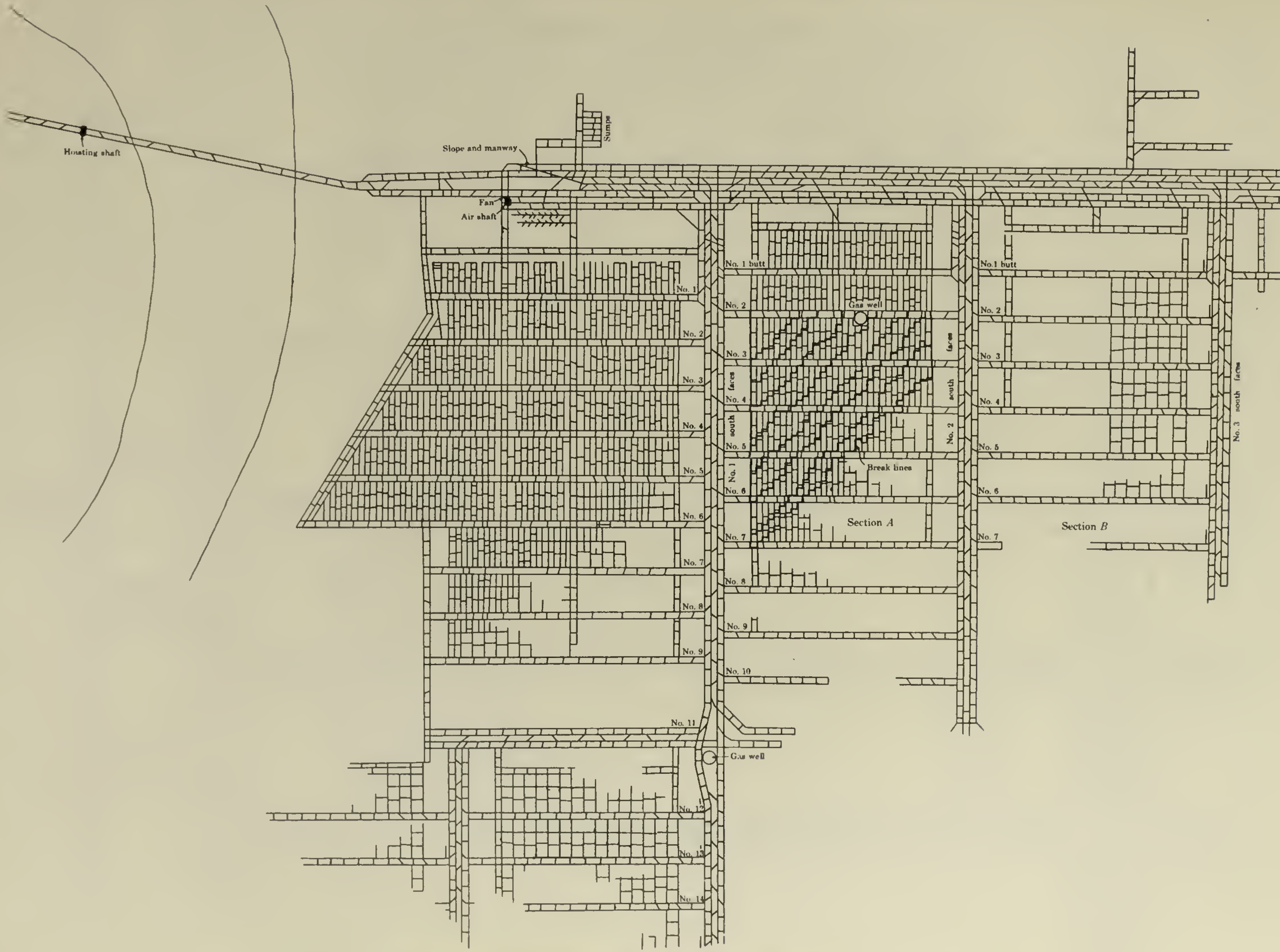

FIGURE 7 . PLAN OF THE MINE DEVELOPMENT. SECTION A AND B SHOWN IN FIGURES 8 AND 9 RESPECTIVELY

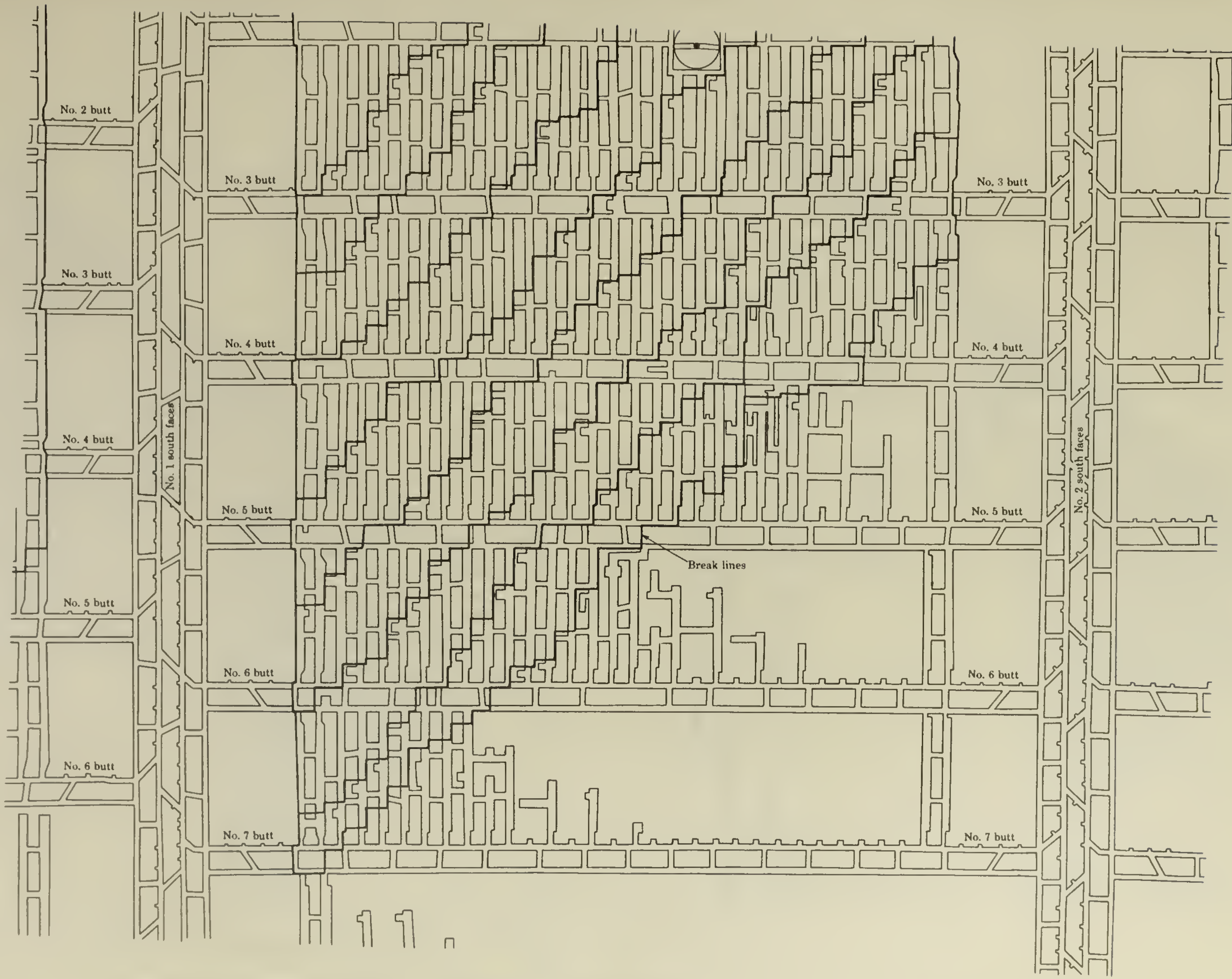

FIGURE 8 - PLAN OF SECTION A OF FIGURE 7. SHOWING WORKED-OUT PANELS AND POSITION OF THE BREAK LINE CROSSING FOUR PAIRS OF BUTT ENTRIES

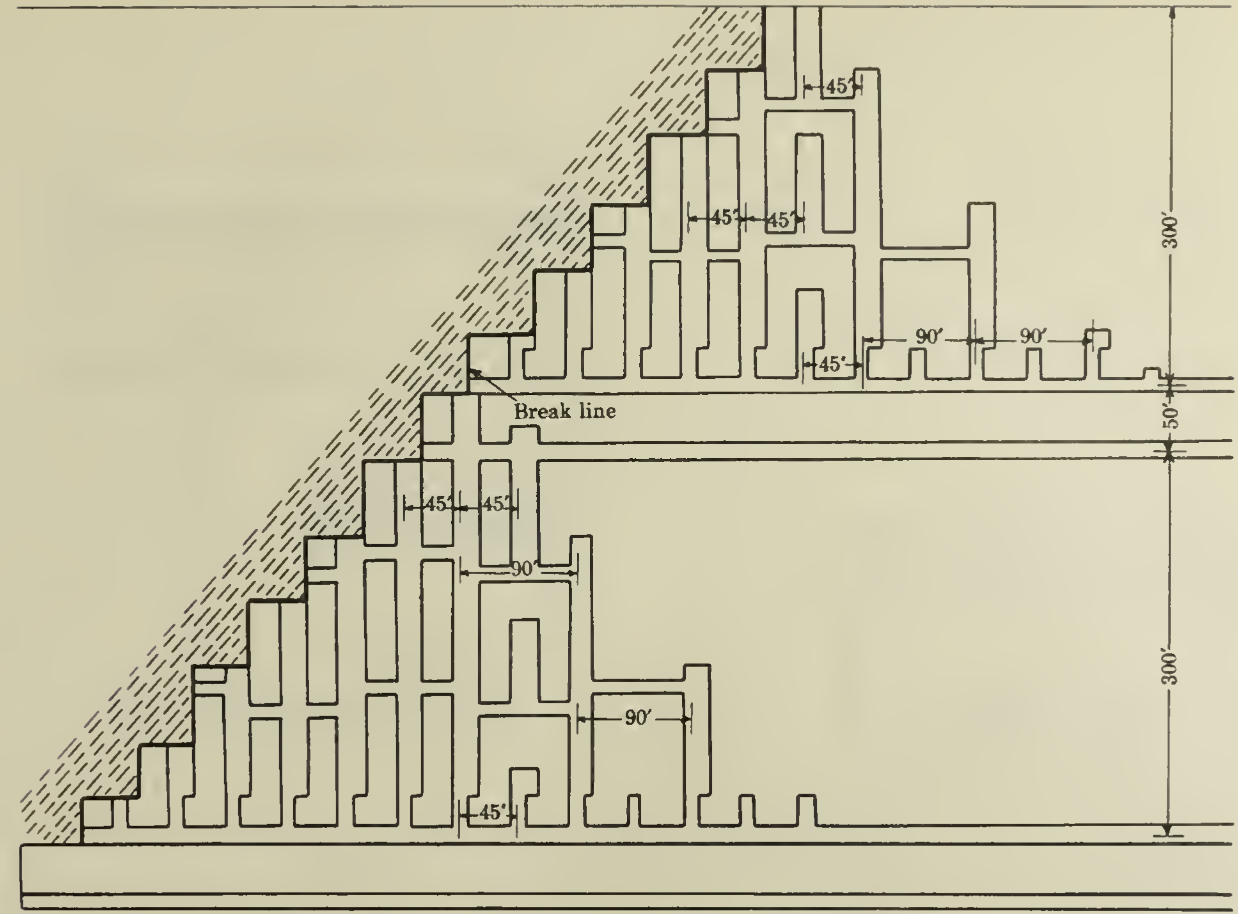

FIGURE 9. - METHOD OF TURNING ROOMS ON 90-FOOT CENTERS

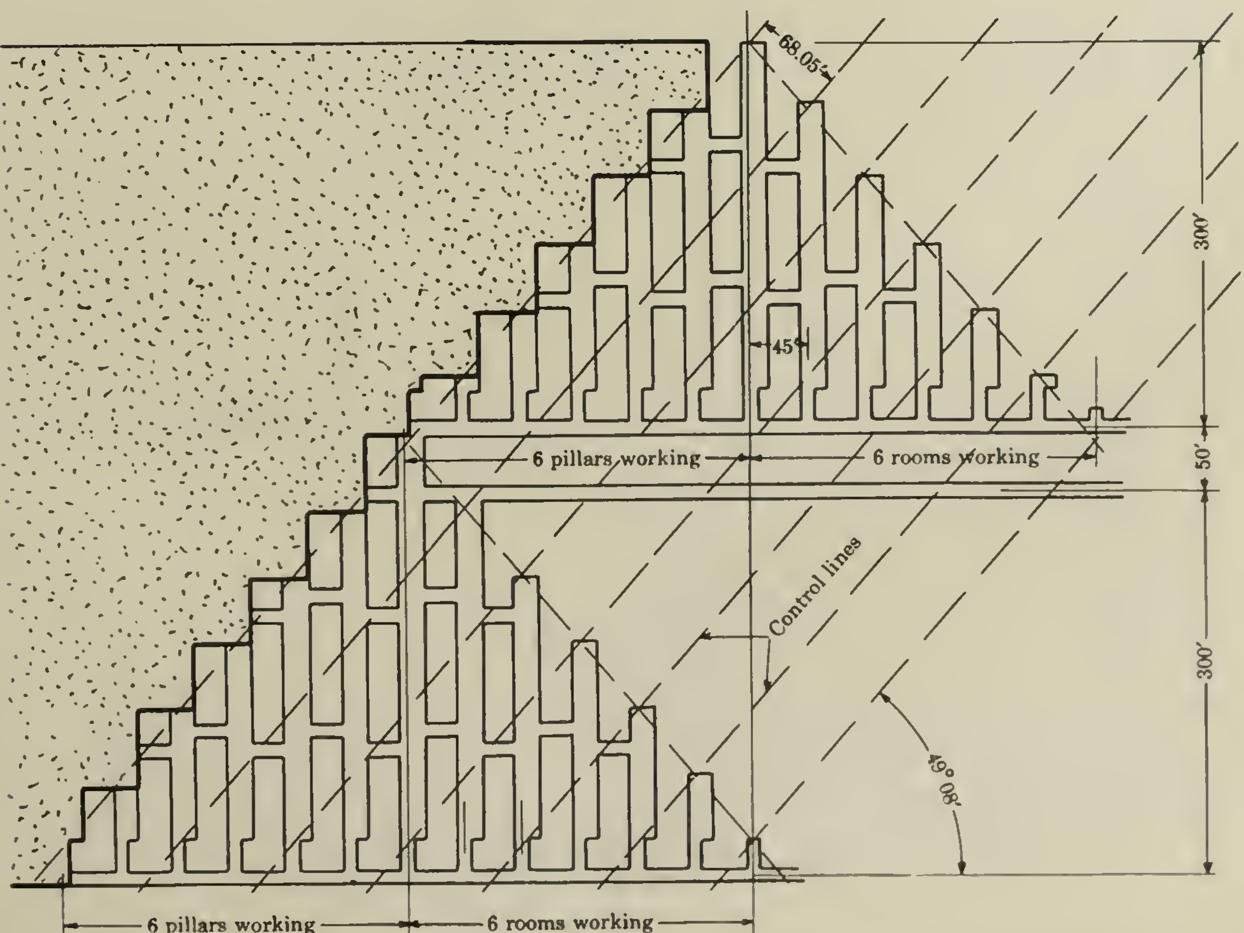

FIGURE 10. - CONTROL LINES FOR PROGRESS OF ROOM ADVANCE AND PILLAR EXTRACTION

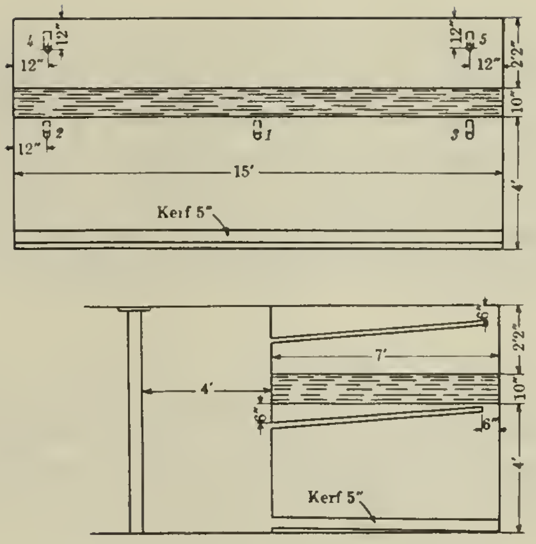

FIGURE 11. - POSITION OF SHOT HOLES IN A ROOM 15 FEET WIDE. FIGURES IN CIRCLE INDICATE SEQUENCE OF FIRING

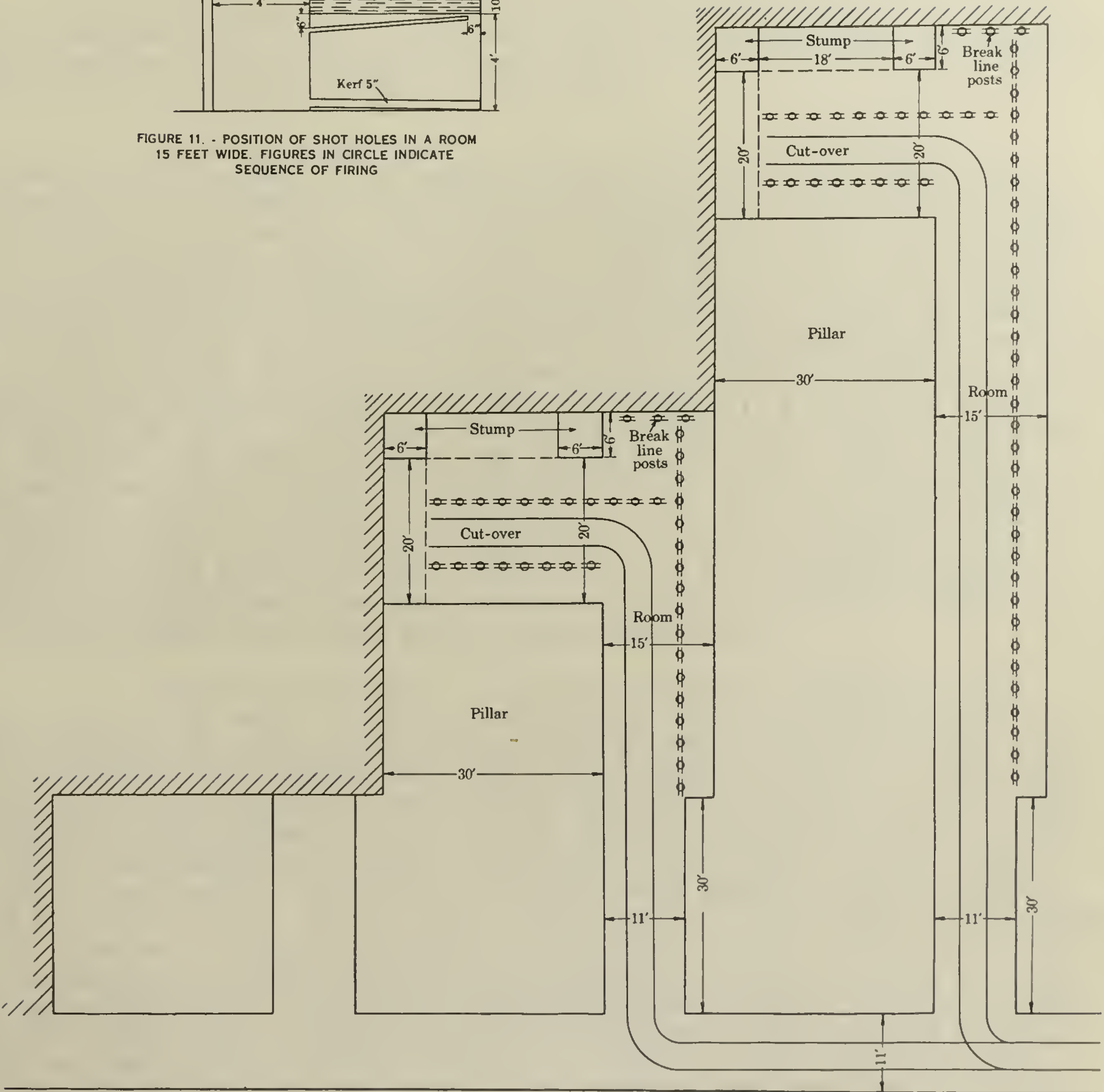

FIGURE 12. - METHOD OF RECOVERING ROOM PILLARS

Control lines are placed on the working map for the guidance of the supervising officials (fig. 10). The distance between the control lines is determined by a line passing through the face of each succeeding room so that the faces will always be 52 feet apart. Strict adherence to the engineering aspects of this scheme is responsible for a recovery of 95 per cent of the coal, a break line under control, and a minimum disturbance of the immediate roof back of the break line, all of which have contributed to an excellent accident record.

(b) (1) Mining. All the coal is mined by electrically driven machines of the shortwall type, ten of which are equipped with 7 feet cutter bars and two with 6 feet bars. These machines average undercutting 240 to 250 tons per shift, and two men are required to operate each machine.

(2) Drilling and blasting. The shot holes for blasting are drilled with post augers by the miner and this is a part of his tonnage rate. The auger is 1-3/4 inches in diameter and the holes are drilled within 6 inches of the depth of the undercut. Five holes are employed for each new cut (fig.11), two in the top about 12 inches from the roof and 12 inches from the ribs and inclined so the back of the hole will be 5 to 6 inches below the roof. In the bottom coal, three shots are employed just below the band of bony, one about 12 inches from each rib and one in the center, and the holes are drilled level. Each shot is charged with two sticks of permissible explosive (1-1/8 by 8 inches), tamped to the collar with clay, and fired electrically by a shotfirer at any time during the shift. The center shot in the bottom coal is fired first, then the two bottom rib shots, and after the bottom coal has been loaded, the two top shots are fired. The bony is separated from the coal by the miner and temporarily stored for future disposal. In a recent month during which 52,458 tons were produced, 9,831 pounds of explosive were used, an average of 5.336 tons per pound of explosive. Since all the product of this mine is used for stoker feed boilers and pulverized combustion, there is no special object in obtaining prepared sizes other than for cleaning purposes.

(3) Rooms and pillars. Rooms are developed by advancing on the face cleats of the coal; in withdrawing pillars, cuts are made through the pillar on the butt cleats which are not so pronounced as the face cleats. The cuts made in the room pillar are 20 feet wide, leaving a curtain of coal 6 feet wide near the gob (fig. 12). This curtain, being as long as the width of the room pillar, is withdrawn by cutting out a space in its center leaving a stump 6 feet or less square at each end. These stumps are either recovered by pick mining or are blasted out in order to leave no roof support, in which latter case the coal is lost. Figure 12 gives the details of the sequence of the operation. When two of these cuts have been taken out, the props are withdrawn back to the breaker line of props and the roof caves to this line, leaving a triangular space, the base of which is 45 feet and altitude of 52 feet measured along the rib of the room. The hypothenuse of this triangle makes an angle of $49^\circ\ 08'$ with the line of the butt entry, as shown in Figure 10. By using this angle along a line to intersect the center of the faces of the rooms, the room faces are kept 52 feet ahead or in the rear of the adjoining rooms. By intersecting the pillar control line with the room control line on the center of the face of

any one room, the next control line will be where it crosses the center of the face of the next room and 52 feet ahead or behind it. The distance between these succeeding control lines in this case works out to be 68.05 feet.

As in all efficiently operated mines, the development entries are kept well ahead of the panel mining. In narrow work no yardage is paid as such, but an additional price per ton is paid the miner and the cutters over the scale paid in wide work, such as rooms.

(4) Loading. All loading of coal and refuse is done by hand. The refuse consists of the bony partingin the coal and is separated by the miner. In narrow places it is loaded and sent to the shaft for disposal. In the room-and-pillars work, the bony is stored in the mine as gob material. At the surface plant the product is sized and hand picked.

(5) Deadwork. In addition to loading coal, the miner must do other work which is covered in the compensation for drilling, charging, loading and cleaning coal, as is the general practice in American coal mines. This involves care of the roof and its support in his working place, the laying of track in rooms and of temporary track in entries. All fallen material that obstructs operations is removed by special daymen. The bony in the coal is gobbed by the miner in the rooms and loaded into cars in entries.

A certain amount of exploration work is necessary when a washed-out area locally called a "faulted" area is encountered. This is where the upper bench of the coal has disappeared, leaving the bony and lower bench in its normal development. The cannel slate decreases in thickness from 3 feet down to 8 or 10 inches; on top of this and immediately under the Mahoning sandstone appears a bed of fine-grained sandstone, very hard, 2 to 3 feet thick, and locally termed "cement stone." A section of the mine under exploration at the time of this presentation covered a face entry with six pairs of butt entries and furnished the largest item of expense under deadwork, labor and material, amounting to 1.463 per cent of the total cost of production.

(6) Timbering. As the immediate roof; a cannel shale varying from 3 to 4 feet in thickness and having few slip planes, affords an exceptionally good roof, artificial supports for this along the entries and haulageways are necessary only at intersections of entries or in wide places and at places where the upper bench of coal is absent. Such supports as are used consist of 6 or 12 inch steel I-beams supported by hitches cut in the ribs of the entries and overlaid with 5 by 6 inch timber lagging; also, at intersections concrete pillars are used. (see fig. 13.)

A definite system of roof support has been adopted for the room-and-pillar work. In rooms 15 feet wide, a row of posts with cap pieces is placed 10-1/2 feet from the rib on the track side, spaced on 3 feet centers and installed within 6 feet of the face of the room before the coal is blasted (fig. 14). A safety post is used under the bony parting while the bottom coal is being loaded, Figure 15. Where crossbars are needed in rooms or advancing entries, 40 to 60 pound T-rails are used. Wooden crossbars are being gradually discarded. In the cutovers in pillar work a row of posts is placed on either

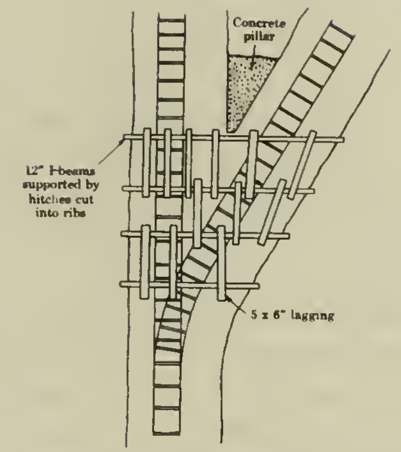

FIGURE 13. PLAN OF METHOD OF ROOF SUPPORT AT INTERSECTION OF ENTRIES

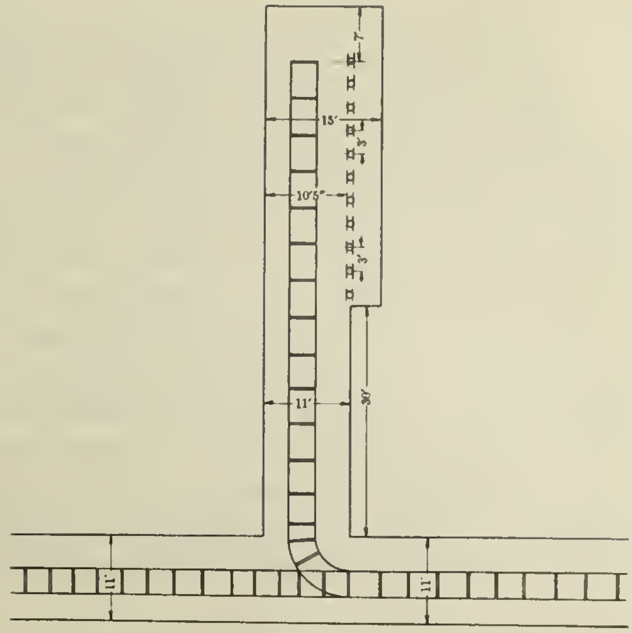

FIGURE 14. - POSITION OF POSTS IN ROOM 15 FEET WIDE BEFORE THE COAL IS UNDERCUT

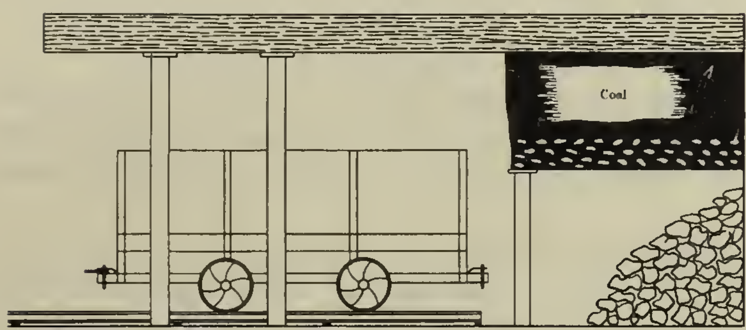

FIGURE 15. POSITION OF SAFETY POST WHILE THE BOTTOM COAL IS BEING LOADED

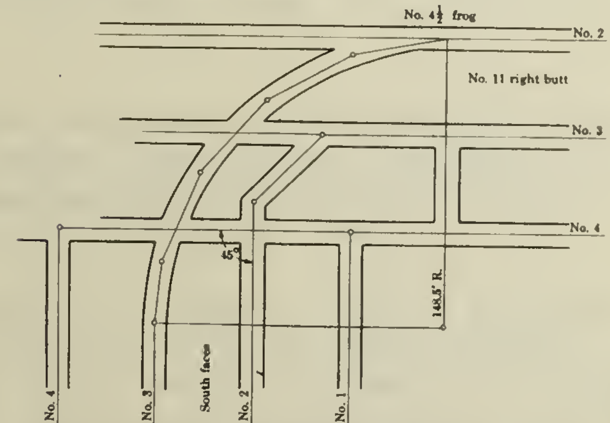

FIGURE 16. - DETAILS OF CURVE LEADING FROM A BUTT TO A FACE ENTRY

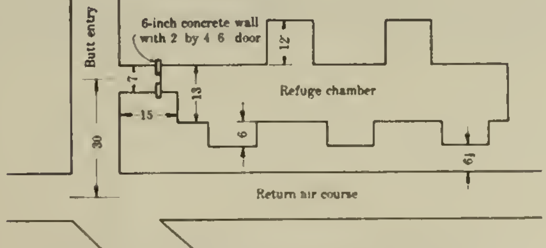

FIGURE 17. - PLAN OF REFUGE CHAMBER AT EACH BUTT ENTRY IN NEW DEVELOPMENT WORK

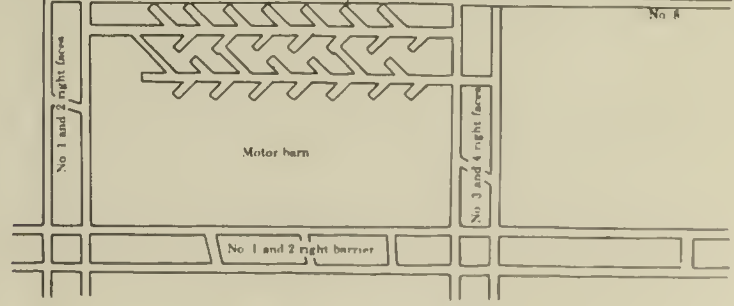

FIGURE 18. PLAN OF MOTOR BARN

side of the track and on 3-foot centers. While cap pieces are used on all posts, there is no uniformity in their dimensions, and the cap is placed in any direction owing to the uniform texture of the cannel shale. When it is desired to make a fall of the roof material in the pillar work, three props with cap pieces are set in a row, sometimes five posts with a common cap piece 6 by 6 inches are used (fig. 12). To bring about a fall of the roof material, the posts are withdrawn by means of a cable and an electric locomotive, and it is estimated that 30 per cent of the posts are recovered. In the year 1927, the quantity of timber used for roof support amounted to 352,390 linear feet and based upon a production of 622,242 tons of coal for that year there were 1,765 tons per linear foot of timber used. Considering the 8 and 10 feet props alone, there were 14.74 tons produced per prop used. In addition, there were used 963 crossbars 6 by 6 inches, varying from 10 to 14 feet in length. The cost data available on timber used from February to September, 1928, inclusive, show labor and material costs of .734 and 1.083 per cent, respectively, of the total cost of production.

Haulage.- Mine tracks and hauling equipment have been standardized. A track gauge of 44 inches is used. The standard radii of curves and switches are 148.5 feet for main haulage with a No. 4-1/2 frog, (fig. 16) and 25 feet for room turnouts with a No. 1-3/4 frog.

Rails are 60-pound on main haulageways, 40-pound on butt entries, and 30-pound in rooms. The main haulageway is double-tracked over a majority of the distance and has a grade of 2 per cent against the load for 1,400 feet near the shaft; the rest of the haul is slightly against the load.

Room tracks are placed and maintained by the miners, and all turns and switches are laid by daymen. Ties 6 by 6 inches by 6 feet on about 18-inch centers are used under 60 and 40 pound steel rails, and at intervals of 5 to 10 feet steel ties are placed. These main-line tracks are ballasted with crushed slate and sandstone. Steel ties are used under 30-pound rails in rooms and temporary track in entries, spaced about 3 feet apart. Permanent track in entries is maintained not less than 200 feet from the face. In all places in the mine a minimum clearance of 4 feet from the side of the car is maintained on one side.

The cars are delivered to and from the faces by locomotives, of which there are eight storage-battery, and four 8-ton reel-type gathering locomotives. The following table shows an average day's work as performed on November 1, 1928;

Average day's work performed by locomotives

Type of locomotive	Shift	No. of cars coal	No. of cars refuse	District operating
Reel	Night	66	4	1st south mains.
Reel	Day	80	2	1st south mains.
Reel	Night	56	9	0 south mains.
Storage battery	Day	57	2	No. 6 butt, 1st south mains.
Storage battery	Day	61	-	No. 7 butt, 1st south mains.
Storage battery	Day	73	-	No. 7 butt, 1st south mains.
Storage battery	Day	69	1	No. 9 butt, 1st south mains.
Reel	Day	81	3	0 south mains.
Reel	Day	72	3	No. 12 and 14 butts, 1st south mains.
Storage battery	Day	54	1	No. 2 south mains.
Reel	Day	91	3	No. 3 south mains.
Reel	Night	62	4	No. 3 south mains.
Total		822	32	

The average number of cars gathered per gathering locomotive on this average day was 71.17. The average number of cars gathered per reel locomotive was 76.57. The average number of cars gathered por storage-battery locomotive was 63.6.

The above table shows that 854 cars were delivered to and from the faces on this average day, 822 of which were coal and 32 were refuse such as bony or cannel shale. On being gathered from the faces the loaded cars are deposited at or near No. 1 room on the butt entries off No. 1 south faces, where they are picked up by the flatroad motor. The average length of haul for the gathering locomotives is from 500 to 1,500 feet.

On the 1st south faces, a flat-road 15-ton locomotive hauls from near No. 1 room on the butt entries to a side track between No. 1 butt and the main east entries. This is the only secondary haulage in operation at this time. On No. 2 and No. 3 south main face entries, the gathering locomotives deliver to a side track between No. 1 butt and the main east entries, as in No. 1 south face entry.

From the side tracks to the shaft is the main-line haul on which three 15-ton Goodman locomotives are employed hauling 20 car trips. The following table gives the number of cars hauled by each locomotive on November 1, 1928, which would be an average day's operation:

Locomotive..........	No. 8	No. 11	No. 16	No. 17	Total
Loads to shaft......	83	339	350	279	1051
Empties from shaft..	20	583	59	287	949

The cars delivered to and from the main-haul side tracks were as follows:

1st South Face

Locomotive.....	No. 8	No. 11	No. 16	No. 17	Total
Loads..........	83	299	270	-	652
Empties........	20	560	59	39	678

2nd South Face

Locomotive.....	-	-	No. 16	No. 17	Total
Loads..........	-	-	40	120	160
Empties........	-	-	-	57	57

3rd South Face

Locomotive.....	-	No. 11	No. 16	No. 17	Total
Loads..........	-	40	40	159	279
Empties........	-	23	-	191	214

No. 8 locomotive makes only two trips to the shaft, one at the beginning and one at the end of the shift, and in the meantime operates on the flat-road haul of No. 1 south face entries. Nos. 11, 16, and 17 locomotives are the main-haul locomotives.

During this shift these locomotives hauled 968 loads to the shaft and 929 empties from the shaft, an average of 322.66 loads, or approximately 1,162 tons and 309.66 empties per main-line locomotive. Although the tables for the gathering locomotives show that 854 loads were gathered during this shift, and the main-haul tables show 1,050 loads taken to the shaft, this difference is due to the fact that the day preceding was an idle day and some cars had been loaded and held in storage to be taken to the shaft on the work day. The difference between the number of loads and empties is due to standing cars at the shaft.

Main haulage.- Table No. 1 has been compiled showing the ton-miles of work accomplished by main haulage locomotives on Nov. 1, 1928, which has been used as an average day for this study. The average weight of coal in the car for 1927, when 173,147 cars were dumped at the shaft with a total tonnage of 622,242, was 3.59 tons. The weight of the empty car is 3,970 pounds or 1.98 tons. The length of the main haul from the 1st south faces is 1.08 miles, and 652 loads and 678 empties were hauled, accomplishing 3,929 and 1,452 ton-miles of work, respectively, or a total of 5,381 ton-miles of work. The length of the main haul from the 2nd south faces is 1.48 miles. One hundred and sixty loads and 57 empties were hauled, accomplishing 1,321 and 167 ton-miles of work, respectively, or a total of 1,488 ton-miles of work.

The length of the main haul from the 3rd south faces is 1.84 miles; 239 loads and 214 empties were hauled, accomplishing 2,568 and 779 ton-miles of work, respectively, or a total of 3,347 ton-miles of work. This shows that a total of 1,051 loads were hauled to the shaft and 949 empties were hauled from the shaft, performing 10,216 ton-miles of work.

Table 1.- Details of ton-miles of electric locomotives on the main haulage, Nov. 1, 1928

(Average weight coal, 3.59 tons; weight of car, 1.98 tons)

	Locomotive 8					Locomotive 11					Locomotive 15					Locomotive 17					Total cars	Total ton-miles
	Cars	Ton-miles load	Ton-miles car	Ton-miles empty	Total ton-miles	Cars	Ton-miles load	Ton-miles car	Ton-miles empty	Total ton-miles	Cars	Ton-miles load	Ton-miles car	Ton-miles empty	Total ton-miles	Cars	Ton-miles load	Ton-miles car	Ton-miles empty	Total ton-miles		
1st South Faces: (Length of haul - 1.08 miles)																						
Loads	83	322	.	-	-	299	1,160	-	-	-	270	1,050	-	-	-	-	-	-	-	-	652	2,532
Cars.	83	-	177	-	-	299	-	640	-	-	270	-	580	-	-	-	-	-	-	-	652	1,397
Total . . .	-	-	-	-	499	-	-	-	-	1,800	-	-	-	-	1,630	-	-	-	-	-		3,929
Empties	20	-	-	43	-	560	-	-	1,200	-	59	-	-	126	-	39	-	-	83	-	678	1,452
Total ton-miles work.	-	-	-	-	542	-	-	-	-	3,000	-	-	-	-	1,756	-	-	-	-	83		5,381
2nd South Faces: (Length of haul - 1.48 miles)	No.8 locomotive makes only two trips to the shaft, one at the beginning and one at the end of the shift. In the meantime it works as flat-road motor on the 1st south faces (see data on secondary haulage, Table 2).																					
Loads						-	-	-	-	-	40	213	-	-	-	120	639	-	-	-	160	852
Cars.						-	-	-	-	-	40	-	117	-	-	120	-	352	-	-	160	469
Total						-	-	-	-	-	-	-	-	-	330	-	-	-	-	991		1,321
Empties						-	-	-	-	-	-	-	-	-	-	57	-	-	167	-	57	167
Total ton-miles work.						-	-	-	-	-	-	-	-	-	330	-	-	-	-	1,158		1,488
3rd South Faces: (Length of haul - 1.84 miles)																						
Loads	.			-	-	40	284	-	-	-	40	284	-	-	-	159	1,131	-	-	-	239	1,699
Cars.	.			-	-	40	-	145	-	-	40	-	145	-	-	159	-	579	-	-	239	869
Total . . .	.			-	-	-	-	-	-	429	-	-	-	-	429	-	-	-	-	1,710		2,568
Empties	.			-	-	23	-	-	84	-	-	-	-	-	-	191	-	-	695	-	214	799
Total ton-miles work.	.			-	-	-	-	-	-	513	-	-	-	-	429	-	-	-	-	2,405		3,367
Total loads hauled to shaft. . .	83			-	-	399	-	-	-	-	350	-	-	-	-	279	-	-	-	-	1,051	
Total empties hauled from shaft	20			-	-	583	-	-	-	-	59	-	-	-	-	287	-	-	-	-	949	
Total ton-miles of work	-	-	-	-	542	-	-	-	-	3,513	-	-	-	-	2,515	-	-	-	-	3,646	10,216	

Table 2.- Ton-miles of work performed on secondary haulage No. 1 south faces and its butts, Locomotive 8, Nov. 1, 1928.

	Cars	Ton-miles load	Ton-miles car	Total load ton-miles	Ton-miles empties	Total ton-miles
No. 5 and 6 Butts:						
(Length of haul 0.39 miles)						
Load	59	82.60				
Car	59		45.55			
Total	59			128.16		
Empties.	59				45.55	
Total ton-miles work . . .						173.72
No. 7 Butt:						
(Length of haul 0.45 miles)						
Load	61	98.54				
Car	61		54.35			
Total	61			152.89		
Empties.	61				54.35	
Total ton-miles work . . .						207.24
No. 8 Butt:						
(Length of haul 0.56 miles)						
Load	73	146.75				
Car	73		80.94			
Total	73			227.70		
Empties.	73				80.94	
Total ton-miles work . . .						308.64
No. 9 Butt:						
(Length of haul 0.63 miles)						
Load	70	158.31				
Car	70		87.31			
Total	70			245.63		
Empties.	70				87.31	
Total ton-miles work . . .						332.95
No. 12 and 14 Butts:						
(Length of haul 0.95 miles)						
Load	75	255.78				
Car	75		141.07			
Total	75			396.86		
Empties.	75				141.07	
Total ton-miles work . . .						537.93

Table 2.- Ton-miles of work performed on secondary haulage No. 1 south faces and its butts, Locomotive 8, Nov. 1, 1928 (Continued).

	Cars	Ton-miles load	Ton-miles car	Total load ton-miles	Ton-miles empties	Total ton-miles
No. 1 South Faces:						
(Length of haul 1.02 miles)						
Load	82	300.26				
Car	82		165.60			
Total	82			465.87		
Empties.	82				165.60	
Total ton-miles work						631.48
No. 0 South Faces:						
(Length of haul 1.25 miles)						
Load	84	376.95				
Car	84		207.90			
Total	84			584.85		
Empties.	84				207.90	
Total ton-miles work . . .						792.75
Total loads hauled	504					
Total empties hauled . . .	504					
Total ton-miles of work. .						2984.71

Averageweight of coal, 3.59 tons
Average weight of car, 1.98 tons
Total, 5.57 tons

As before mentioned, No. 8 locomotive makes only two trips to the shaft, one at the beginning and one at the end of each shift. In the meantime this locomotive works as the secondary haulage locomotive on the 1st south faces and accomplishes 2984.71 ton-miles, which with its 542 ton-miles on the main haulage totals 3516.71 ton-miles.

The table shows that No. 17 locomotive, which hauls an even trip during the shift, accomplishes 3,646 ton-miles of work, while No. 11 and No. 16 locomotives accomplished 3,513 and 2,515 ton-miles of work, respectively. The total ton-miles accomplished in the shift were 13,190.7 and the coal hauled to the shaft totaled 3,773 tons. This is the equivalent to an average haul per ton of 3.50 miles, and since the car capacity averaged 3.59 tons, the average haul of each 3.59 tons on main and secondary haulage was 0.97 miles.

On Nov. 7, 1927, the average haul from the face to the shaft was 9,177 feet. On June 30, 1926, the average haul from the face to the shaft was 8,665 feet, showing an increase of 512 feet during this period.

Mine Cars.- There are 740 mine cars of 100 cubic feet capacity at this plant with steel sides and ends and wooden bottoms, carrying 3.59 tons of coal as an average load. The axles are 3 inches in diameter and run in plain bearings.

The wheels are 18 inches in diameter, run loose on the axle, and are equipped with ratchet brakes. The wheel base is 32 inches.

During 1927, in 156 working days 173,147 cars of coal and 12,538 cars of refuse were dumped at the shaft, an average of 1,190 cars per working day or a turn around on 740 mine cars of 1.608.

Drainage pumps - type and capacity.- Excepting for the tunnels under the river this mine could be called dry. The mine water is gathered into two main sumps on the north side of the main east entries by twelve 2-inch and five 4-inch gathering pumps. From these sumps the water is pumped to the surface through the manway by two 10-inch pumps. At the present time it makes about 360 gallons of water per minute, the pumping generally being done on the night shift. The labor and material cost for drainage is 1.021 and .254 per cent, respectively, of the total cost of production.

Ventilation.- Ventilation is supplied by a 12 by 6 foot fan, belt-driven by a 200 hp. motor, exhausting 300,000 cubic feet per minute against a 2-inch water gauge. There is a good supply of air at each working face. Line brattices are used where necessary to conduct the ventilation to the working face, and doors constructed of lumber and canvas are used in preference to swing brattices. Permanent stoppings built of concrete are put in place as the entries advance. Overcasts built of concrete, reinforced with steel, are put in where required. A notable feature at this mine is that few doors are used on haulageways, overcasts being employed instead. The air courses are kept clean and free from fallen material.

The tunnels under the river and the shaft bottom are ventilated by a system independent of the main mine ventilation. A 6-foot disc fan placed at the head of the manway blows 32,000 cubic feet per minute against a 1-inch water gauge and is belt-driven by a 15 hp. motor which is used for this purpose. By this method the dust from the dump at the bottom of the shaft passes up the shaft, which is the outlet or return for this ventilating system. By this arrangement the shaft is kept free from ice in winter weather.

Power, source of supply, kind, and voltage.- Power is purchased in 2,300-volt A.C. units. At the mine side it is reduced to 250 volts D.C. by two motor generator sets of 300-kw. capacity each. The voltage used underground is 250 D.C. During September, 1928, 6.3 kw.-hrs. were consumed per ton of coal mined.

Tabulation of data.- Tables have been made up on classification of labor, and operating cost percentages.

The first, classification of labor, gives the percentage of men employed in the various classes of labor; such as contract labor, inside daymen, haulagemen, and outside day-labor.

The second gives the operating cost percentages under the following items:

Mining.- Cost of labor and supplies for loading, machine mining, company coal, mining machine repairs, miscellaneous mining expense, and supplies for these items.

Deadwork.- Cost of labor and supplies, for grading, cleaning falls, outside disposal of refuse, miscellaneous, and driving entries in faulted territory.

Timbering.- Cost of labor and supplies for timbering.

Ventilation.- Cost of operating the ventilating system and repairs to equipment.

Drainage.- Cost of operating the drainage system and repairs to equipment.

Haulage.- Cost of gathering and hauling coal, operating and maintaining mine roads or tracks, mine cars, and haulage system in general.

Tipple.- Cost of operation and repairs to equipment.

Power.- Cost of substation operation and repairs to equipment.

Purchased power.- Cost of power purchased from a public utility.

Repairs to structures.- Cost of repairs to the various structures at the plant.

Mine office expense.- Under mine office expense are included superintendence, mine foreman, engineering, mine office expenses, storeroom expenses, welfare work, first-aid and mine rescue, patrolmen and police, miscellaneous mine expense, machine shop expense, supplies and repairs to equipment.

Insurance, liability and compensation.- Cost of workmen's compensation insurance premiums.

Classification of Labor

Inside:	Per cent	Outside:	Per cent
Loaders	62.82	Labor-foreman . . .	0.26
Cutters	4.72	Clerks	0.52
Contract labor.	67.54	Garbageman.	0.26
Inside Daymen:		Laborers.	2.63
Mine foreman.	0.26	Electricians. . . .	0.26
Assistant mine foreman.	0.78	Substationmen . . .	0.26
Firebosses.	1.31	Blacksmiths	0.52
Roadmen	3.15	Car-repairers . . .	0.52
Day laborers.	5.24	Carpenters.	0.26
Bratticemen	0.52	Tippleman	1.83
Wiremen	0.78	Machinist	0.26
Pumpers	0.78	Watchmen	0.78
Drillmen.	0.52	Supply clerk. . . .	0.26
Machine repairmen . . .	0.78	Total outside	
	14.12	day labor	8.62

Haulage:	Per cent
Motor boss	0.26
Motormen	4.72
Motor brakemen	4.72
	9.70
Total inside labor . .	91.36

Operating Cost Percentages February to September, 1928, Inc.

Tonnage 428,667.50; 53,583 tons av. mo.

	Labor	Material	Total, per cent
Mining	63.000	1.254	64.254
Dead work.	1.193	.280	1.473
Timbering.	.734	1.083	1.817
Ventilation.	1.672	.448	2.120
Drainage	1.021	.254	1.275
Haulage.	10.110	3.722	13.832
Tipple	1.618	.302	1.920
Power.	.291	.032	.323
Purchased power. . . .	–	5.936	5.936
Repair to structures .	.058	.086	.144
Mine office expense. .	4.797	.326	5.123
Insurance liability and compensation. .	1.783	–	1.783
Totals	86.277	13.723	100.000

Efficiency Data

Cutting machines, 2 men on crew Production per shift 240 to 250 tons

Tons produced per pound of explosive used	5.336 tons
Tons of coal produced per linear foot of timber used	1.765 tons
Tons of coal produced per prop used.	14.74 tons
Average number of cars gathered per gathering locomotive	71.17
Average number of cars gathered per storage-battery locomotive	63.6
Average number of cars gathered per reel locomotive.	76.57
Average number of cars handled to and from working face by gathering locomotive.	854

Average length of haul for gathering locomotive ranges from 500 to 1,500 feet.

Main haulage, total ton-miles shift (empty and loaded cars)	10,216
Secondary haulage, total ton-miles per shift (empty and loaded cars).	2,984.7
Total ton-miles per shift.	13,190.7
Tonnage of coal delivered to hoisting shaft per shift.	3,773
Average length of haul on main and secondary haulage.	0.97 miles
Number of times each car was loaded per shift.	1.608

Accidents.- During the year 1927, in 156 working days 622,242 tons of coal were produced, 384 men were employed, and there were 20 lost-time accidents. A total of 629 days was lost due to these accidents, which are attributed to the following causes: Falls of roof and coal 9, or 45 per cent; transportation 3, or 15 per cent; timbering 3, or 15 per cent, and miscellaneous causes 5, or 25 per cent. The following table shows tons produced per accident:

Tons produced per accident

Cause	No. of accidents	Total days lost	Tons per accident
Falls of roof and coal . .	9	313	69,138
Transportation	3	49	207,414
Timbering.	3	57	207,414
Miscellaneous.	5	210	124,448.4
Total.	20	629	31,112

Safety.- This mine is termed a gassy mine: therefore, electric safety cap lamps are used by the employees. The supervising officials use a flame safety lamp. At important points along the haulageway 250-volt electric lights are used, for which the power is taken from the trolley line.

Inspection.- Fire bosses are employed who examine each working place once before the men go on shift, and once while they are on shift, for any dangerous condition, and who report their findings in the daily report book.

The working places are also visited twice each day by the assistant mine foreman of the district, who examines for any dangerous condition and instructs the miner in his work.

Blasting.- Permissible explosives are used, fired electrically by a shotfirer, who examines the place thoroughly before firing a shot. Clay tamping is used, and the cuttings are scraped from under the mining before blasting.

Rock dust.- The mine has been rock-dusted several times from the face to the shaft, and rarely was a working place seen where rock-dust was more than 30 feet from the face. During 1927, 4,560 sacks of rock-dust were used. Barriers and shelves loaded with rock-dust have been placed at various points in the air courses. It is now the practice after dusting air courses to place platforms loaded with rock-dust at intervals of 100 feet. To minimize dust in the shaft, the loaded cars are sprinkled by an overhead device as they approach the hoisting shaft.

Recovering posts.- When posts are being recovered in making a fall in pillar work, they are pulled out by a locomotive with a piece of cable, thereby eliminating a chance of injury to workmen.

Trolley lines.- The trolley lines are well guarded at all important places by means of boards supported by anchors in the roof and forming an inverted trough.

Tracks.- The tracks are well kept, and all over the mine a clearance of 4 feet is provided on one side. Points of switches and frogs are blocked and guarded to prevent anyone from getting his feet caught in them. All switches are provided with parallel throw switch stands that lie close to the floor.

Manways.- Manways are provided so that the miner on leaving the butt entries does not have to travel on the haulageways.

Telephones.- Telephones are placed at important points on the main haulageways and at each alternate butt entry.

First aid.- A quantity of first-aid material, with stretcher and blankets, is kept at various points in the mine for immediate use in case of an accident to an employee.

Refuge chamber.- At each pair of butt entries in all new development work, a refuge chamber as shown in Figure 17 has been constructed. This chamber is kept clean and well posted and has a concrete wall built near the mouth with a door ready to be put in place if in case of a fire or explosion it is necessary for men to barricade themselves.

Fire extinguishers.- An 80-gallon chemical fire extinguisher, mounted on a truck with 300 feet of hose ready to be used in case of fire, is kept in the motor barn. At each butt entry and at important places about the mine, 2 1/2-gallon chemical fire extinguishers are maintained and kept ready for use.

Motor barn.- A motor barn with the stalls concreted has been built near the main east entries. Figure 18 is a plan of the barn, which is equipped with apparatus for charging storage-battery locomotives, and with a shop and motor pit for repair work.

SUMMARY

The features at this mine that are especially notable are as follows:

1. There seems to be a place for everything, and everything in place. The mechanical equipment is modern and is kept in good condition.

2. The mining method, the long break lines, and the method of roof control are giving good results. The practice of removing the entry stumps and butt entry chain pillars by extending the room through these pillars fits in well with the maintenance of long break lines.

3. The haulage is mechanical throughout; the cars are delivered to and from the working faces by locomotives. Tracks are kept clean and in good condition.

4. While approximately 360 gallons of water per minute are being pumped, in general, the mine may be called dry. A good portion of this water is seepage from the river.

5. Roof support is supplied where necessary on the haulageways. Steel I-beams are used for crossbars. The method of posting in rooms and pillars is adequate for the needs of the roof; however, some improvement could be made by applying a good cap piece, 18 inches long, 4 inches thick and wide enough to cover the post. The practice now being followed in using T-rails instead of wooden crossbars in rooms and pillars should give good results.

6. Ventilation is good at all places. Overcasts and stoppings are well built and maintained; line brattices are carried to the face where necessary.

7. Supervision at this mine is good; each working place is visited at least three times during the shift by a supervising official.

UNDERGROUND MANAGEMENT AND PAY SYSTEM USED

All underground employees are under the direct control of the mine foremen and their work is supervised by assistant foremen assigned to different districts. The foreman and his assistants are paid on a day or contract rate, and no bonus is paid to any employee. The machinemen and miners only are paid on a tonnage rate. A higher tonnage rate is paid for work in narrow places to miners and machine men, but not as a bonus. No particular task to constitute a shift's work is assigned any of the underground employees.

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CIRCULAR 6120 MAY, 1929

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

GEOPHYSICAL ABSTRACTS

NO. 1

BY

FREDERICK W. LEE

This paper is the first of a contemplated series which will contain abstracts of current articles and publications dealing with applied geophysics. The abstracts will be prepared, for the most part, by officials and engineers of mining and exploration companies, in cooperation with the United States Bureau of Mines. It is believed that useful and timely information dealing with the science of applied geophysics can thus be adequately presented. The Bureau plans, if possible, to procure the original papers from which these abstracts are prepared and to assist those who may be interested in obtaining translations or photostat copies.

Scott Turner

SCOTT TURNER,
Director.

Circular No. 6120.
May, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

GEOPHYSICAL ABSTRACTS [1]

No. 1

Compiled by Frederick W. Lee [2]

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Ayvazoglou, W., U. S. Bureau of Mines, Department of Commerce, Washington, D. C.
Barton, Dr. D. C., Petroleum Building, Houston, Tex.
Eve, Dr. A. S., McGill University, Montreal, Canada.
Foote, Dr. Paul D., Mellon Institute, Pittsburgh, Pa.
Gish, Dr. O. H., Carnegie Institution, Broad Branch Road, Washington, D. C.
Hartley, Kenneth, Humble Oil Co., Houston, Tex.
Hutchinson, Prof. W. Spencer, Massachusetts Institute of Technology, Cambridge, Mass.
Jenny, Dr. W. P., 210 McGowen Avenue, Houston, Tex.
Karcher, Dr. J. C., Geophysical Research Corporation, 65 Broadway, New York City.
Keys, Dr. D. A., McGill University, Montreal, Canada.
Knappen, Dr. R. S., Gypsy Oil Co., Tulsa, Okla.
Lane, Prof. Alfred C., Tufts College, Boston, Mass.
Lee, Dr. F. W., Bureau of Mines, Department of Commerce, Washington, D. C.
Leonardon, E. C., 25 Broadway, New York City.
McLaughlin, Prof. Donald H., Harvard University, Cambridge, Mass.
Ostrand, Dr. C. E. Van, Interior Building, Washington, D. C.
Roman, Dr. I., Geophysical Research Corporation, 65 Broadway, New York City.
Ruark, Dr. A. E., Mellon Institute, Pittsburgh, Pa.
Scholl, Louis A., Box 1805, Houston, Tex.
Sundberg, Dr. Karl, Gulf Refining Co., Houston, Tex.
Truemann, O. H., Humble Oil Co., Houston, Tex.
Weaver, Paul, Drawer C, Houston, Tex.
Weelden, Dr. A. von, Shell Petroleum Corporation, Dallas, Tex.
Wright, Dr. F. E., Carnegie Institute, Washington, D. C.
Zuschlag, Dr. Theodor, Swedish American Prospecting Co., 26 Beaver St., N.Y.City.

1 - The Bureau of Mines will welcome reprinting of this article but requests that the following footnote acknowledgment be used: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"
2 - Supervising engineer, geophysical section, U. S. Bureau of Mines.

1 - GRAVITATIONAL METHODS.

WIRELESS CONTROL OF COINCIDENCE OUTFIT FOR RELATIVE GRAVITY MEASUREMENTS

By Berger and Brückner

Zeitschrift fuer Instrumentenkunde, vol. 48, 1928, p. 366

A detailed description is given of radio apparatus for controlling a coincidence outfit for relative gravity measurements. This is probably of little interest, since many types of apparatus could be used. The general method employed is the interesting thing about this paper. As long ago as 1883 von Sterneck made a proposal to control two coincidence boxes with the same clock, and many persons have suggested the use of radio for this purpose. In accomplishing this it would appear to the reviewer that this method is probably inferior to that used by the Marland Oil Co., but Berger and Brückner must be given the credit for important advances in the coincidence method. In their apparatus a clock at a central station actuates a relay, which operates a short-wave transmitter (40 m.). At the receiving station the amplification is accomplished at low frequencies. The signal operates a relay which moves a slitted screen in a coincidence apparatus. This screen has been improved considerably, to decrease the mean error in observing a coincidence. As usual, a telescope is used to observe the flashes of light. It has a scale in its focal plane and positions of successive flashes are easily read off and recorded by a second observer. This makes possible a higher accuracy in determining the exact instant at which coincidence would have occurred at the zero point on this scale.

This method obviously depends on the constance of the mechanical and electrical lag in both sending and receiving apparatus, and may be adversely criticized from this standpoint. -- A. E. Ruark.

GRAPHIC METHODS FOR CORRECTION OF GRAVITATIONAL OBSERVATIONS WITH RESPECT TO TOPOGRAPHY, AS WELL AS WITH RESPECT TO THE SUBTERRANEAN MASSES

By B. V. Numerov

Bulletin of the Geological Committee, appendix to vol. 44, No. 1, Leningrad, 1925.

Deducing briefly the formulas for the establishment of corrections of observations in connection with the topographic influence as proposed by Hayford and others, the author explains the difficulties of determination of the "gravitational anomalies" by this method and proposes to solve this problem, to great advantage, by using a map provided with contour lines, as such a solution may be attained in a geometrical way.

The solution of the problem of the influence of subterranean masses by a geometrical method is explained by the author also. -- W. Ayvazoglou.

THEORETICAL CASES FOR THE USE OF GRAVITATIONAL (GRAVIMETRIC) METHODS IN GEOLOGY

By B. V. Numerov

Bulletin of the Geological Committee, vol. 44, No. 3,
Leningrad, 1925.

The author examines the problem of applying gravitational (gravimetric) methods in the study of geological formations of superstrata.

According to the author this problem may be approached by examining special examples of forms of underground strata and by calculating their influenc on the gradients and on the gravity.

The following forms are examined in the article: that of a finite parallelopiped, an infinite vertical parallelopiped, and oblique rectangular prism and an oblique triangular prism.

The article gives formulas by which the sensibility of variometers and pendulums for the case of an infinite horizontal stratum may be compared.

The influence of such a stratum on the gradients, measured by a variometer, depends on the relation of the depths, while the gravity is characterized by the thickness of the stratum.

Based on the formulas, a series of tables is prepared with the aid of which the character of the changes of the gradients for any infinite prism of any position with regard to the horizontal plane may be calculated easily. -- W. Ayvazoglou.

THE INTERPRETATION OF GRAVITATIONAL OBSERVATIONS

By B. Numerov

Bulletin of the Institute of Astronomy, No. 15, Leningrad, Feb. 15, 1927

Three methods for the construction of a gravitation curve of an infinite prism of any section are examined in the article. The first method is a graphic one and consists of counting the number of sectors included within a given section. The second method is based on calculations. In the third method a special instrument, a kind of planimeter, is used.

The reciprocal problem of the interpretation of the observed gravitational curve is proposed to be solved by the method of successive approximations. -- W. Ayvazoglou.

CORRECTION OF OBSERVATIONS MADE BY MEANS OF A GRAVITATIONAL VARIOMETER WITH RESPECT TO TOPOGRAPHY

By B. Numerov

Bulletin of the Institute of Astronomy, No. 17, Leningrad, December, 1927

This paper is an exposition of an analytical and graphic method of taking topographic conditions into account when effectuating observations with the help of an Eötvös torsion balance. The integration is carried out in cylindrical coordinates. The analytical method allows the integration with respect to height to be fulfilled with the utmost accuracy; the integration with respect to the radius is carried out by means of special formulas of quadratures, taking into consideration the peculiar character of the changes of the functions being integrated. The integration with respect to the angle is produced with due accuracy on the basis of the expansion of functions under the sign of the integral into Fourier's series.

The graphic method allows the determination of the effect of topographic conditions within the limits of the radius from 50 to 500 m. and over that, with the help of graphics dividing the area into sectors of equal effect. Thus the determination of the effect of the relief leads to the calculation of the area occupied by the mass lying between two adjacent horizontals. -- W. Ayvazoglou.

GRAVIMETRIC RESEARCHES OF FERRUGINOUS QUARTZITES IN THE REGION OF KRIVOY ROG

By P. Nikiforov, S. Ghirin, A. Terentiev, and N. Veshniakov

Bulletin of the Institute of Practical Geophysics,
No. 3, Leningrad, 1927, pp. 322-385

The gravimetric prospecting described in the article was carried out in the years 1925-26.

Chapter I

In addition to the description of the work itself practical features for the organization of the work are given also. The researches took place on three plots shown on the annexed map of the region. The problems of the research were as follows:

1. Location of ferruginous quartzite beds.

2. Determination of the elements of the deposit (extent, angle of inclination, magnitude).

3. Separating the ore-bearing zones and contouring the lenses.

4. Checking of the supposed faults.

Chapter II

Concerns the technical arrangement of the observations; gives description and preparation of the instruments and their distribution for the work.

Chapter III

Works out the data obtained by observation.

Chapter IV

Shows the calculated results by means of the following maps and graphics:

1. Topographical map with contours.

2. Curves of the horizontal gradient in the direction of the research lines.

3. Diagrams of the vectors of the entire horizontal component of the gravity force and curves of equal intensity of the force of gravity.

4. Graphics of the disturbing values of curves. -- W. Ayvazoglou.

2 - MAGNETIC METHODS

EXPERIMENTS ON THE MAGNETIZATION OF ROCKS

By F. Loewinson-Lessing and V. Mitkevich

Bulletin of the Geological Committee, vol. 44, No. 5, Leningrad, 1925

The idea of starting these experiments arose in connection with the magnetic anomaly observed in the volcanic group of Karadagh in Crimea.

Based on this anomaly the authors believed that the magnetization of these rocks is caused not by the terrestrial magnetism but by the atmospheric electrical discharges.

The well-known fact that the lightning which produces fulgurites in the rocks produces at the same time permanent magnetization could not be applied here, as there were no fulgurites in the Karadagh rocks; thus the question as to whether a permanent magnetization of the rocks from a distance is possible or not was to be solved.

A series of artificial magnetizations of the rocks by means of a straight electromagnet, with an iron rod 400 mm. in length and 50 mm. in diameter, was performed by the authors. A magnetic field of 3,500 gausses was used.

More than 200 samples of rocks and minerals were submitted to magnetometric tests before and after the experiments.

The rocks and minerals used for the experiments, as well as the results obtained, are given in a table.

They are arranged according to special groups, and the results are summarized by comparison of the data obtained.

Plutonic Rocks: 59 per cent of the samples tested did not display any natural magnetization, but 68.8 per cent of them became magnetized artificially and retained the magnetization permanently. Most granites were free from natural magnetization but were susceptible to artificial magnetization.

Effusive Rocks: All effusive rocks, even those magnetized by natural magnetization, may be magnetized artificially.

Contact-Metamorphic Rocks: Of the rock in the contact-metamorphic group 55.5 per cent are free of natural magnetization, but in most cases they can be magnetized sufficiently strongly.

Metamorphic Rocks: Of the metamorphic rocks 61 per cent did not show any magnetization; an intense artificial magnetization was attained for rocks with weak magnetization.

Sedimentary Rocks: Rocks classed as sedimentary are mostly free of magnetization and, in general, can not be magnetized.

Based on the experiments, the authors drew the conclusion that the permanent magnetization displayed by the rocks must in some cases be attributed to the discharges of lightning and not to terrestrial magnetism. The question as to whether there are means to distinguish the magnetization of the rocks produced by the terrestrial field from that produced by lightning was raised. The authors answer this question in the affirmative and believe that they have found the method by which the residual magnetism produced by the lightning can be determined. At the same time they express the desire that their method be applied in the study of the permanent magnetization of rocks. -- W. Ayvazoglou.

THE COMPASS DIAL WITH OSCILLATING NEEDLES
(The Inclined Compass Dial)

By V. Pavlinov

Bulletin of the Geological Committee, appendix to vol. 46, No. 7, Leningrad, 1927

The article gives a detailed description of the compass dial with oscillating needles used at the present time in prospecting.

The author explains the rules necessary for taking care of the dial and insists on the necessity of studying them, as even the smallest damage to the dial, which usually can not be repaired by emergency means, makes the dial useless, thus increasing the obstacles for prospecting.

THEORIE UND PRAXIS DER MAGNETISCHEN SCHUERFMETHODEN

By Johann B. Ostermaier

Internationale Zeitschrift fuer Bohrtechnik, Erdoelbergbau und Geologie, vol. 37, No. 1, 1929, pp. 1-3

This article is a continuation of a series started in 1927, Nos. 10, 11, 13, and 14. The author describes the construction and properties of the Kohlrausch variometer and the Bidlingmaier double compass. Interesting is the determination of temperature variations by means of thermometers of the same caloric capacity as the magnetic system or, in other words, by means of thermometers the variations of which do not lead or lag compared with the temperature variations of the magnet system. This is accomplished by using thermometers of different dimensions. Small thermometers are leading, while large thermometers are lagging in regard to the temperature variations of the magnet system. -- T. Zuschlag.

THE INFLUENCE OF TOPOGRAPHY ON THE EARTH MAGNETIC VERTICAL FIELD

By J. Koenigsberger

Sonderdruck aus Gerlands Beiträge zur Geophysik, vol. 20, No. 3/4, 1928. Akademische Verlagsgesellschaft m. b. h., Leipzig.

The unevenness of the terrain causes a topographic effect on the magnetic vertical intensity Z, which is difficult to deduce theoretically. An extremal positive and negative value for ΔZ (the additional vertical intensity) for a certain susceptibility, K, apparently is $\Delta Z = \pm 4\pi KZ$. This effect for $K \leq 0.01$ is almost proportional to K. The largest difference between summits (+) and steep valleys (-) observed in mountains of gneissic rocks (Bellinzona, Ticino, Switzerland) was about 120 γ. The vertical intensity in the immediate neighborhood of the wall of the quarry was about -40 γ. In case of a very small K, the effect observed was equal to zero. Thus, the effect depends on the angle of the slope and should depend on the situation of the wall to the magnetic meridian; but this latter dependence could not be observed. Therefore, by the topographic effect the value of the susceptibility of large masses of rock in the earth magnetic field can be calculated approximately. The observations were made with a vertical variometer which gives an accuracy of about $\pm$ 3 γ for the results. The local inhomogeneity of the places where the topographic effect was studied was about $\pm$ 4 γ. -- W. Ayvazoglou.

INTERPRETATION OF CHARTS OF MAGNETIC ISANOMAL CURVES AND PROFILES (WITH 9 FIGURES).

By J. Koenigsberger

Sonderdruck aus Gerlands Beiträge zur Geophysik, vol. 19, No. 2, 1928.
Akademische Verlagsgesellschaft m. b. h. Leipzig.

The isanomal curves for the vertical and horizontal intensity and for the inclination with respect to the surface of the earth, based on the exact theory of induction, are given by formulas and figures for a sphere and for an oblate ellipsoid of revolution; it is also shown how the depth of their centers under the surface of the earth can be calculated. The formulas are given for the isanomal curves of the vertical component which are caused by an elongated ellipsoid of revolution with vertical axis and an oblate ellipsoid of revolution in different positions. The characteristic parameters can be found in tables with the aid of which - and by interpolation, also, for some curves - the center depth and, in case of comparatively small depth, the form and position of the ellipsoid also can be found.

The conclusions drawn from magnetic charts of vertical intensity anomalies are as follows: The magnetic anomalies show the depths of the center of the disturbing mass from the surface to be between 0.2 to 20 km.; comparatively often depths of from 3 to 6 km. were found. Conducting channels of masses belonging to effusive rocks on the surface have a depth of from 3 to 6 km.; their continuation down to the plastic zone is dipping under gentle angles. The higher zone, partially of effusive type, of these basic rocks has a depth of center from 300 to 1000 m., and the same is approximately true for some so-called plutonic rocks, which therefore do not always continue in great depths. The great anomalies marked on maps with scale of 1:1,000,000 or more have very varying depths of center of from 0.3 to 30 km. -- W. Ayvazoglou.

APPARATUS FOR CALIBRATION OF MAGNETOMETERS

By W. Pavlinoff

Bulletin of the Institute of Practical Geophysics of the Supreme Council of Public Economy, No. 2, Leningrad, 1926, pp. 177-183

This paper deals with the task of constructing an instrument capable of producing easily and with precision a homogeneous horizontal and vertical magnetic field of a desired intensity within the bounds of a sphere 30 cm. in diameter.

The apparatus constructed by the author represents a system of two magnetic shells in Helmholz's arrangement.

The diameter of the instrument is about 7 feet and the number of windings in both coils together is 158.

The instrument can be placed so that its axis coincides with the vertical line, or the axis can be placed horizontally in every azimith. The electric current is measured by Weston's amperemeter of high sensitivity. Any magnetometer can be examined by this instrument and its curves can be constructed. The characteristic curves of the Tiberg-Thalen and Thomson-Thalen magnetometers are given in the appendix to the article. -- W. Ayvazoglou.

THE MAGNETIC FIELD OF BODIES OF REGULAR SHAPE FROM THE VIEWPOINT OF MAGNETOMETRIC STUDIES

By J. Bahurin

Bulletin of the Institute of Practical Geophysics, No.2, Leningrad, 1926, pp.1-63
Bulletin of the Institute of Practical Geophysics, No.3, Leningrad, 1927, pp.148-255

In Chapter I the author examines the magnetic field of an oblate ellipsoid of revolution magnetized in the direction of its axis of revolution. Diagrams showing the vectors of total intensity and the lines of equal values of horizontal and vertical intensities produced by the ellipsoid, as well as tables for the calculation of the data of vertical and horizontal intensities for any ellipsoid of revolution, are given.

Chapter II considers the magnetic field of an infinitely long elliptical cylinder. The situation of the characteristic points of the field and their relation to the dimensions and to the depth of the ore body are established by detailed mathematical analysis.

The article is completed by many tables and diagrams.-- W. Ayvazoglou.

MAGNETIC PROSPECTING IN THE IRON ORE REGION OF THE PROVINCE OF TULA

By N. Rose

Bulletin of the Institute of Practical Geophysics, No.3, Leningrad, 1927, pp.137-147

The object of the magnetic work carried out by the author consisted of establishing the influence of the deposits of weak magnetic limonite on a magnetic field at the surface; the thickness of the horizontal ore layers of the region is about 36 m. and the depth is from 21 to 27.6 meters. The magnetic surveying was carried out by means of Moureaux-Chasselon's magnetic theodolite and Dover's needle indicator.

Two courses of 16 points each with an average distance between them of about 640 m. were made. Deviations from the normal values of the magnetic elements in the point of intersection of the ore fields by the course are given in tables and diagrams. -- W. Ayvazoglou.

3 - SEISMIC METHODS

THE SEISMOGRAPH IN THE GULF COAST

By Mark C. Malamphy

The Oil Weekly, Jan. 18, 1929, pp. 31-34

A brief review of the history of seismographic exploration in Texas is followed by an elementary description of the fundamental ideas involved in the method, the principles on which the different types of seismographs are based, and a statement of the main physical facts which determine the velocity and path of an explosion wave through the ground. This is followed by an outline of the method of working and of interpreting the results, illustrated by a diagram and graphs of the time-distance curve. These graphs are entirely imaginary and represent what the writer thinks should happen under the assumed conditions according to the still popular "Mintrop theory;" no other method of interpretation is mentioned, so that the reader is left to infer that these ideas are universally accepted. The last half of the article is devoted to a description of field methods, as used for the finding of "salt domes," the organization of field parties, estimates of cost of such exploration, difficulties of operation, and many practical details which should be very valuable to any company contemplating such work. -- Kenneth Hartley.

4 - ELECTRICAL METHODS

CONCERNING SOME PHENOMENA OBSERVED DURING THE ELECTRICAL METHOD OF PROSPECTING

By F. Shkliarevsky

Bulletin of the Geological Committee, appendix to vol.44, No. 2, Leningrad, 1925.

This article describes prospecting by the electrical method, in particular the method of equipotential lines, using the linear electrodes.

Copper pyrites and the sericite schists deposited in greenstone ores were prospected. The author gives explanation of some phenomena by which the conditions of work were made difficult; he mentions especially the fact that an absolute damping of the sound in the telephone of the feeding circuit could not be obtained. According to his opinion the main reasons for this were:

1. The damping was prevented by the induction of circuits in the neighborhood of the electrodes and of the feeder connecting the generator with the electrodes.

2. A phase displacement caused by self-induction.

3. The inductional humming produced in the neighborhood of the pyrites.

The author concludes that there were other phenomena observed during the electrical prospecting which could not be explained satisfactorily by him, thus the cooperation of expert electrophysicists for solving these phenomena is very desirable. -- W. Ayvazoglou.

RADIO IN ORE PROSPECTING

By A. Petrowsky

Bulletin of the Institute of Practical Geophysics, No. 1, Leningrad, 1925, pp. 135-151

Contents of the article:

1. First tests with radio for prospecting purposes (Trustedt, Lowery and Leimbach).

2. The shade or absorption method.

3. The beam or reflection method.

4. The interference or superposition method.

5. The return method or quarter-wave.

6. The wave measuring method.

7. The measuring of short waves.

8. The present state of these methods.

The methods are illustrated by plans showing the cross sections of the terrain. -- W. Ayvazoglou.

ELECTRIC CONDUCTIVITY OF ORES AND ROCKS

By D. Murashov, E. Berengarten, A. Etcheistova and L. Khudiakova

Geological Committee. Series of applied Geophysics and Prospecting. No. 1, Leningrad, 1928.

The present paper contains the summarized results of a series of measurements of the electric conductivity of various ores and rocks from a number of deposits, in respect to which the question has arisen, or may arise, as to the applicability to them of the methods of electrical prospecting.

Methods of measurement are described. The results obtained from the investigation of the specific resistance of 180 samples, as well as the mineralogic and petrographic composition of the latter, are shown in the appended tables.

The samples characterize various deposits of the Ural, Caucasus, South Russia, Kazakstan, Altai, and Central Siberia.

Fully amenable to the successful application of methods of electric surveying are various more or less continuous pyritic copper and polymetallic ores with prevalence of pyrite, chalcopyrite, bornite, chalcocite, galenite, and arsenopyrite, enclosed in metamorphic schists, limestones, and other rocks. -- W. Ayvazoglou.

ERDOEL IN MECKLENBURG-SCHWERIN (GERMANY)

By

Internationale Zeitschrift fuer Bohrtechnik, Erdoelbergbau, und Geologie, vol. 37, No. 2, 1929, p. 16

The Luebthen-Jessenitz dome was investigated by seismic and electric methods after a dry hole northeast of the dome had been abandoned at a depth of 650 m. According to the electric investigations of Piepmeyer and Co., oil should be expected at 1,000 to 1,200 m. depth on the northwestern part of the dome. The Mecklenburg Landtag has provided the money for the drilling operations to check up this prediction. Near the village of Oberg five producers of 10 to 30 tons per day were brought in at a depth of 510 to 550 m. -- T. Zuschlag.

OIL IN MECKLENBURG-SCHWERIN

By H. Rantenkranz, Celle.

The salt dome of Luebthen-Jessenitz has been known for several decades already.

In 1927 a drill for oil was abandoned because at a depth of 650 m. there was still found younger Tertiary. With seismographs and electric methods the Northeast and Northwest boundaries of the salt dome have now been accurately determined, and a new drill has been decided. According to the electric indication the oil would occur at a depth of 1,000 to 2,000 m. Another occurrence of oil has been lately discovered near Oberg. Five drills have reached the oil horizon at a depth of 510 to 550 m. and yield a daily production of 10 to 30 tons. The thickness of 10 to 15 m. of the oil horizon seems to promise a continuous production. -- W. P. Jenny.

ELECTROMETRIC METHODS IN ORE PROSPECTING AND EXPERIMENTAL INVESTIGATIONS AT RIDDER'S MINE DURING THE SUMMER OF 1924

By A. Petrowsky

Bulletin of the Institute of Practical Geophysics,No.1, Leningrad, 1925,pp.107-132

The author describes the results of the detailed investigations made at Ridder's Mine (Altai). The method of the measurement of the natural direct currents, as well as that of the measurement of the artificial direct currents (Kelly's method), with a few modifications introduced by the author, were used. The modifications concern the method of moving the electrodes and of adding resistance. Petrowsky's method of introducing the current into the earth by means of a great number of small electrodes placed along the boundary of the investigated field and connected with each other by wire on a state of equal potential (called by him the "Dot system of arranging electrodes") takes rank between Schlumberg's and Lundberg's methods. According to the author it was used with great advantage.

The article is illustrated by pictures and diagrams. -- W. Ayvazoglou.

NATURAL ELECTRIC FIELD PRODUCED BY ORE

By A. Petrowsky

Bulletin of the Institute of Practical Geophysics,No.1, Leningrad, 1925,pp.87-104

The following questions are examined in the article:

1. The natural circumstances and conditions by which a natural electric field is produced by ore.
2. The basis of the theoretical examination of this question.
3. Differential equations and limitations in the case of an infinite medium.
4. The expression for the electric potential and potential gradient in this case.
5. A polarized sphere in a medium limited on one side by a plane.
6. The final equation of the electric potential and the potential gradient.
7. The transformation of the formulas.
8. The examination of an electric field along a plane of polarization created by an ore body.
9. The examination of an electric field along a plane perpendicular to the plane of polarization.

The article is provided with the following diagrams:

1. The electric currents produced by the ore body.
2. Vertical and horizontal sections of a polarized sphere.

3. Electric field produced by the polarized field.
4. Polarized sphere and its electrical representation.
5. Distribution of the electric power along the axis of the abscissas.
6. Distribution of the electric power along the axis of the ordinates.-- W. Ayvazoglou.

DETERMINATION OF THE LOCATION, DEPTH AND THICKNESS OF A SPHERICAL ORE BODY BY OBSERVING THE EARTH CURRENT PRODUCED

By A. Petrowsky

Bulletin of the Institute of Practical Geophysics, No.3, Leningrad, 1927,pp.3-36.

Contents:

Chapter I

1. The problem and the formulas obtained by its analysis.
2. The general nature of the electric profile in the plane of polarization.
3. The zero curves.
4. The maximum curves.
5. The general nature of an electro-profile which is obtained in a plane perpendicular to the plane of polarization.
6. The zero curves.
7. The maximum curves.

Chapter II

1. The determination of the location, depth, and thickness of a spheric ore body.
2. Graphic calculation.
3. Analytic calculation.
4. Graphic interpretation.

Chapter III

1. The formulas which express the gradients of the potential along a line that is inclined to the plane of polarization.
2. The general nature of an electroprofile in a plane that is inclined to the plane of polarization.
3. The zero curves.
4. The maximum curves.
5. The full scheme of the observation and calculation of the elements that characterize the spheric body.

The article is illustrated by many diagrams. -- W. Ayvazoglou.

CALCULATIONS OF AN ARTIFICAL ELECTRIC FIELD

By A. Petrowsky

Bulletin of the Institute of Practical Geophysics,No.3, Leningrad, 1927, pp.39-63.

Contents of the article:

1. The relation between the charge and strength of the current in a three-dimensional system.
2. The Schlumberger field.
3. The Petrowsky field.
4. The Lundberg field.
5. The equipotential lines and current lines of the field.
6. Procedure of the calculation of potential values.
7. Application of the special tables.
8. Procedure of the calculation of equipotential lines.
9. Procedure of the calculation of potential gradient.

The procedures of the calculations of (6) and (9) as well as the application of special tables (7), are given in the form of examples. -- W. Ayvazoglou.

THEORY OF THE MEASUREMENTS OF EARTH CURRENTS

By A. Petrowsky

Bulletin of the Institute of Practical Geophysics, No.1, Leningrad,1925, pp.73-87.

Contents:

1. The problem (what appearance have equipotential surfaces and lines of force when conducting into the earth a receiving electrode of spheric shape?).
2. Differential equations and limitations of the influence of an electrode in an unlimited space.
3. A general integral for an E-potential.
4. The meaning of the arbitrary constant.
5. The influence of a flat limited surface upon the E-potential.
6. The meaning of the E-potential in this case.
7. The potential of a spheric electrode half buried in the ground.
8. The distribution of the equipotential surfaces and of the current lines with different conditions.

By the aid of the formulas calculated, a series of graphs showing the electric field near an electrode is calculated and drawn. -- W. Ayvazoglou.

ELECTROMETRIC INVESTIGATION OF THE UPPER-ARSHINSK (URAL) ORE BED, ACCOMPLISHED IN SUMMER OF 1926.

By A. Petrowsky, R. Skariatin, and L. Kleiman

Bulletin of the Institute of Practical Geophysics, No.3, Leningrad,1927, pp.64-86.

The chief problem consisted in stating to what extent the results expected from the electric method of investigation may agree with the results of an ordinary prospecting.

The work consisted of:

1. Surveying of isolines by means of an alternating current.
2. Surveying of gradients by means of direct current.
3. Supplementary measurements, such as the resistance of the ground in different parts of the field, distribution of the intensity along the conducting line of the electrode, and voltage drop on principal and auxiliary lines.

A benzine motor coupled with an alternator of high frequency and a direct current dynamo were used as a generating set. The results of the electric investigation were considered favorable, as the data obtained agreed with those procured by direct digging. -- W. Ayvazoglou.

THEORY OF THE RETURN METHOD

By A. Petrowsky

Bulletin of the Institute of Practical Geophysics, No.2, Leningrad,1926,pp.143-173

Contents:

11. The calculation and graphic construction of the coefficient themselves.
12. Supplementary constructions.
13. Calculation and graphic construction of a complete reaction curve for a dipole.
14. Reaction curve for an ideal dipole.
15. Reaction curve for a dipole that has some leakage.
16. Reaction curve for a symmetric and straight conductor.
17. Reaction curves for small coils. -- W. Ayvazoglou.

5 - RADIOACTIVE METHODS

ON THE THEORY OF AN ASPIRATOR DEVICE FOR THE INVESTIGATION OF ORE SAMPLES WITH REGARD TO THEIR RADIO ACTIVITY

By V. J. Baranov

Bulletin of Geological Committee, appendix to vol. 44, No. 4, Leningrad, 1925.

The theory of the device, based on the natural ionization of radioactive bodies is proved by experiments.

A diagram shows curves obtained by theoretical calculations as well as those obtained by experiments.

In addition to the direct purpose, that is of the investigation of radioactivity of ore samples for which the device is constructed, the device may, according to the opinion of the author, be used also for preliminary comparison of the radioactivity of rocks instead of the Elster and Gietel method. -- W. Ayvazoglou.

APPLICATION OF EBERT'S ION-METER FOR A RAPID DETERMINATION OF THE RADIOACTIVITY OF THE SAMPLES ON THE SPOT

By V. J. Baranov

Bulletin of the Geological Committee, appendix to vol. 46, No. 1, Leningrad, 1927.

The author proposes to provide the groups assigned for the research of radioactive deposits with Ebert's ion-meter owing to the importance of the investigation of the ionization of the air in the region of the research work and to the possibility of a rapid determination on the terrain itself of the radioactivity of the samples collected.

A full description of the instrument and its operation is given in the article. -- W. Ayvazoglou.

ASPIRATOR FOR INVESTIGATION OF RADIOACTIVITY
OF THE SAMPLES OF ORES IN GEOLOGICAL COLLECTIONS

By A. P. Kirikov

Bulleting of Geological Committee, appendix to vol. 44,
No. 1, Leningrad, 1925.

The idea of the device is based on Ebert's principle which consists of counting the ions by catching them on a charged electrode placed in a tube through which a fixed volume of air passes. Data characterizing the work of the device are given in a table, and the operation of performing measurements is explained. The device may be used with great advantage for establishing deposits of radioactive ores by testing samples of existing collections. The determination of a relative radioactivity of ores in a box with about 40 samples requires from one to two minutes only. Experiments performed with the aid of the new device were very satisfactory. The apparatus is intended only for investigation of samples and not for a field device. -- W. Ayvazoglou.

6 - GEOTHERMAL METHODS

THE CALCULATION OF THE INFLUENCE OF ROCKS ON THE NATURAL
AND ARTIFICIAL HOMOGENOUS FIELDS IN THE GROUND

(Problems of Geothermics, Earth Magnetism,
and Geoelectrics)

By J. Koenigsberger

Sonderdruck aus Gerlands Beiträge zur Geophysik, vol. 18,
No. $\frac{1}{2}$, 1927. Akademische Verlagsgesellschaft m. b. h. Leipzig.

In addition to the statements made by Maxwell, the author shows us how the mathematical expressions, calculated for the influence of rocks (spheres and ellipsoids of rotation) on a primary homogeneous field of geothermics, earth magnetism, and geoelectrics, can be derived by taking into consideration their borders on the earth surface. The application of general theorems is examined in a special case, that of the deviation of geoelectric current lines on the surface of the earth (Schlumberger's probe method) caused by a sphere and oblate ellipsoid of rotation in the ground. The probe method is compared briefly with the integral method or methods in which all factors enter, and some practical conclusions are derived. -- W. Ayvazoglou.

INTERNAL HEAT OF EARTH IS STUDIED TO ASCERTAIN FACTS ON WHICH TO BASE GEOLOGICAL PRINCIPLES

By C. E. Van Orstrand, Geophysicist, Geological Survey

The United States Daily, Washington, D. C., Feb. 15, 1929.

The question of the internal heat of the earth has been studied almost from the time of the establishment of the U. S. Geological Survey in 1879. In 1920 a report on geothermal data, based generally on the observations made by placing a thermometer in the water flowing from the mouth of a well, was published by N. H. Darton. In this report it was noted that in certain areas in eastern South Dakota the rates at which the temperatures increase from the surface downward vary somewhat uniformly from about 1° F. in 20 feet to approximately 1° F. in 45 feet. But this method of making temperature tests has proved to be not satisfactory, thus the author of this article undertook the task of designing and constructing the apparatus necessary for more accurate measurements. Two different types were developed, one of which was based on the electric resistance thermometer, and the other on the mercury thermometer of the maximum type. It has been necessary to abandon the electric method for the present, owing to the fact that the cable that will meet the requirement of withstanding the dissolving action of oil and salt water must be of relatively large diameter, possibly one-half or three-fourths of an inch; it becomes of such weight and proportions that it can not be manipulated in a deep well without the aid of powerful machinery.

The machine which is being used to-day for lowering mercury thermometers into a well by means of a piano wire consists of a steel frame and reel, a standardized wheel for accurately measuring depths, and a cylindrical cam which distributes the wire on the reel and thus prevents the transmission of impacts to the thermometers as a result of the slipping of the coils of wire on the reel. No power, other than hand power, is needed. It is a remarkable fact that a machine weighing 58 pounds, exclusive of the piano wire, which weighs 2.7 pounds per 1,000 feet, can be used to sound wells to depths of more than 4,500 feet.

Concerning the question of the value of temperature tests the scientists were not yet able to give a definite answer.

It has been assumed ordinarily that during the millions of years of the earth's existence its outer layers have gradually cooled to depths of about 200 miles. In contrast to this hypothesis is the comparatively recent supposition that practically all of the heat of the earth is due to the disintegration of radium. There are at present wide differences of opinion with regard to the causes of the irregular distributions of heat in the outer layers of the earth's crust. Thus, it has long been known that the temperatures at the same depths in different localities are not the same. For example, at Fairmont, West Va., a temperature of 170° F. was found at a depth of 7,500 feet; while at Longmont, Colo., a temperature of 212° F. exists at a depth of only 6,600 feet. In general the rates at which the temperatures increase with the depth vary from the extremely rapid rate of 1° F. in 20 feet (the value found in some of the oil

fields in Wyoming) to 1° F. in 200 feet (gold mines at Johannesburg, South Africa). No serious attempts have been made to explain these variations in the temperature of the rocks at the same depths until the U. S. Geological Survey found in some oil fields in Wyoming and California that the temperatures of the rocks at given depths were higher than the temperatures found at the same depths in the rocks immediately surrounding the fields.

The peculiar distribution of heat existing in these oil domes is attributed to the radioactivity, or, possibly, to the chemical reactions within the oil itself. Other investigators sought the explanation in deepseated intrusive masses, conduction of heat in the rocks, and the migration of waters in deeply buried sands. It is impossible at present to render a final decision as to the merits of all these hypotheses.

Most intensive investigations are conducted to-day in the United States, as a result of cooperation of various organizations, and the author believes that a precise geothermal survey will ultimately provide the facts on which certain fundamental principles of geology may be established. -- F. W. Lee.

7 - UNCLASSIFIED METHODS

GEOPHYSICAL METHODS USED FOR PROSPECTING OF ORE DEPOSITS

By A. K. Gedovius

Bulletin of the Geological Committee, vol. 44, No. 1, Leningrad, 1925.

After giving a brief description of the fundamental purpose of applied geophysics, the author enumerates the geophysical methods used for prospecting of ore deposits. He divides the methods into two groups, according to the physical properties of ore bodies.

In the first group he includes those methods characterized by a direct manifestation of the ore bodies at a certain distance (magnetic method, gravitational method, and radioactive method).

In the second group are the methods by which he examines the physical properties of ore bodies which, although not manifested at a distance, influence the distribution of the electric energy sent into the ground by artificial means (methods based on electric conductivity and dielectrical constant, and the seismic method).

A brief historical review of the development of the methods, as well as a few examples illustrated by diagrams, is given. -- W. Ayvazoglou.

8 - GEOLOGY

NEUE ERDOELVORKOMMEN IN BULGARIEN

By

Internationale Zeitschrift fuer Bohrtechnik, Erdoelbergbau und Geologie, vol. 37, No. 1, 1929, pp. 7.

Indications of oil were found near Tschirpan, Rupki, and Swoboda. Professor Bontschew, Bulgarian geologist, has been asked by the Bulgarian Government to make investigations. -- T. Zuschlag.

ERDGAS BEI SISAK, SHS

By Dr. Lupas Waagen

Internationale Zeitschrift fuer Bohrtechnik, Erdoelbergbau und Geologie, vol. 37, No. 2, 1929, pp. 9-10

The newspaper report that oil was found near Sisak is not true. However, gas was struck while drilling water wells. The author discusses the geology and general structure of the Sisak plain, as well as the special conditions under which the gas was found. -- T. Zuschlag.

DAS ERDOELGEBIET DER HALBINSEL SANTA ELENA (ECUADOR)

By

Internationale Zeitschrift fuer Bohrtechnik, Erdoelbergbau und Geologie, vol. 37, No. 1, 1929, pp. 4-5

The article is an abstract of a geological report by Cunningham Craig. The general geology, structure, the known resources and the methods of production are discussed briefly. -- T. Zuschlag.

DIE ERSTEN ERFOLGREICHEN ERDOELBOHRUNGEN IN BRASILIEN

By

Internationale Zeitschrift fuer Bohrtechnik, Erdoelbergbau und Geologie, vol. 37, No. 3, 1929, p. 22

The Brazilian Ministry of Agriculture announces that the first important wells in Brazil were brought in near Botocatu and Piracicaba in the State of Sao Paulo. Chief Engineer Schemuly investigated several locations in the States of Parana and Sao Paulo by means of his patent Polarisator. His predictions were proved on the Facendas Barro Branco and Maria Isabella near the Rio Iguassu, as

well as on Facenda Araqua near Piracicaba. On Facenda Saltinho near Botocatu on the Rio Tiete he predicted oil in five horizons down to 1,600 m. depth. -- T. Zuschlag.

THE DRILLING ACTIVITY AT BORYSLAW FROM 1924 TO 1928

By Dr. Alfred Pfaff, Soln, near Muenchen

Abstracts from "Petroleum" vol. 25, No. 1.

Detailed tables are given about the drilling activity of the different oil companies at Boryslaw from July 1, 1924, to July 1, 1928. The tables show that the small companies have developed a much larger percentage of drilling activity than the larger concerns, which conclusion would be in favor of a decentralization of the oil industry at Boryslaw.

From 1924 until 1928, there have been drilled at Boryslaw a total of 130.017 m.; i.e. 96,000 m. for new drills at an average cost of 570 zloty per meter, and 34.017 m. for deepenings at an average cost of 1,600 zloty per meter. The daily production in July, 1928, amounted to 148,55 cisterns from a total of 338 wells with an average depth of 1500 m.

The annual production of 54,000 cisterns has been quite constant for the last few years and corresponds to 20.3 completely drilled wells per year. The average total production of a well is therefore 2,660 cisterns, or 83.6 per cent of the total production in 1913, which amounted to 3,182 cisterns.

In order to encourage the drilling activity, and especially the deepening of old wells, a new system of taxes is proposed, which is to increase the taxes with increasing daily production of the well. -- W. P. Jenny.

Bulgaria

In the neighborhood of the town of Tschirpan in Central Bulgaria there have been found very promising indications of oil, which are being examined by Professor Bontchev of the University of Sofia. Beside the indications near Tschirpan, oil has been found under a layer of hard rock at a few meters depth near Roupki and Svoboda, both in the county of Tschirpan. Great interest has been shown in these oil deposits by foreign concerns. -- W. P. Jenny.

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INDEX OF AUTHORS AND SUBJECTS

Circular 6121 May, 1929

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

METHOD AND COST OF MINING ZINC AND LEAD AT MINE NO. 2, TRI-STATE DISTRICT, PICHER, OKLA.

BY

Wm. F. Netzeband

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Circular No.6121.
May, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

METHOD AND COST OF MINING ZINC AND LEAD AT MINE NO. 2 TRI-STATE DISTRICT, PICHER, OKLA.[1]

By Wm. F. Netzeband[2]

INTRODUCTION

The mode of ore occurrence and the methods and costs of mining at one of the zinc-lead mines in the Tri-State zinc and lead district, Oklahoma, are presented in this paper for the information of mine operators in other districts.

HISTORY

No. 2 mine property was acquired by lease in 1918 after some 30 holes had been drilled and a shaft had been sunk to the 305-foot level by the first lessees. After acquisition of the lease, the present operating company drilled a rich ore body on the 260-foot level and in 1920 started sinking two shafts. Production was begun late in 1921, and the mine has been operated more or less continuously to the present time.

The mill has a capacity of 400 tons in 10 hours, the usual length of the mill shift in the district.

GEOLOGY

The geology of Mine No. 2 is very similar to that of Mine No. 1[3] except that the Cherokee shale is thicker, ranging from 80 to 160 feet, and all the formations are correspondingly deeper, the Short Creek oolite horizon here being at a depth of about 300 feet. The oolite has not been found on this property, but the lime comes in at about the level at which the oolite should appear.

ORE DEPOSITS

The ore deposits of Mine No. 2 occur in the brecciated and bowlder ground above the Short Creek oolite. The main or 260-foot level is about 40 feet above the oolite horizon, and the lower or 300-foot level is immediately above it.

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1 The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be made: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"
2 One of the consulting engineers, U. S. Bureau of Mines.
3 Netzeband, Wm. F., Bureau of Mines Information Circular 6113, 1929, 11 pp.

The ore deposition is controlled by a zone of shearing and fracturing, and the mineable areas are divided into two distinct runs which have a northwest trend with a lean or practically barren area dividing them. This characteristic has controlled the mining to a large extent, for it enables the mine superintendent to locate the large pillars in lean ore and leave only small pillars in the richer ore. Figure 1 shows the outlines of the ore body and the pillars.

The ore on the main level occurs disseminated throughout the jasperoid breccia as patches or lenses in the jasperoid, or cementing massive bowlders of jasperoid breccia and chert. The ore of the lower level is more of the bowlder type than that on the main level and occurs chiefly cementing the bowlders but also with some disseminated ore in the jasperoid breccia.

The gangue minerals are jasperoid breccia, chert, and dolomite. Calcite is found in minor quantities. Sphalerite is the principal ore mineral; galena is found only in minor quantities around the edges of the ore body. When galena is encountered in this ore body, the limit of the ore can be expected.

EXPLORATION AND ESTIMATION OF ORE RESERVES

The exploration and estimation of ore reserves at Mine No. 2 are similar in many respects to the methods used at Mine No. 1.[4]

After the mine was opened up and the general character of the ore deposit had been determined, drilling operations were concentrated along the trend of the shear zone. Later drilling has proved what appears to be a parallel shear zone or an offset of the zone now being mined.

The shear zone was very difficult to drill before the water was drained off, and in consequence the cost of drilling was high. The early drilling cost \$1.50 per foot, and many of the holes had to be finished on company time at the rate of \$25 per diem. The present price of drilling is \$1 per foot. On this property 61,080 feet of drilling has been done.

EARLY MINING METHODS

The methods of mining have not changed at this mine; the ore is still loaded by hand into cans; the cans are trammed by mules to the shaft and then hoisted to the surface, dumped into the mill hopper without being detached from the rope, and returned immediately to the shaft bottom.

The mining method, the open stope system with pillar support, has remained the same, but the character of the ground requires the use of larger pillars than at most mines of the district.

4 Netzeband, Wm. F., Bureau of Mines Information Circular 6113, 1929, 11 pp.

FIGURE 1.—Plan of mine No. 2

DEVELOPMENT SYSTEM

The first shaft was sunk by the original company to a depth of 305 feet. Considerable water was encountered, necessitating the installation of large pumps. When the present company took over the property, they drilled out a shallower and more extensive ore body upon which they sank two shafts. The original and deeper shaft has been retained for use as a pump shaft to drain the mine workings.

Main Shafts.- The main shaft was sunk as the mill shaft by the present company. The shaft is 5 by 11 feet in cross section. It was originally planned to install skips, but this has never been done. The shaft was sunk on contract, but no costs are available.

Auxiliary Shafts.- A field shaft was sunk at the time the mill shaft was put down, and the two shafts were connected by a drift on the 260-foot level. This field shaft is of the standard size, 5 by 7 feet in cross section, and is used for handling both men and materials; no men are handled at the mill shaft.

Two "field shafts" were sunk in 1926 on isolated ore bodies and were connected with the mill by surface trams. The sinking of these shafts was contracted for at the rate of $9 per foot in shale and $13 per foot in rock. The contractor furnished all labor and explosives and the company built the sinking derrick, installed the dump track, and furnished all equipment, material, and power.

The wages paid by the contractor were $5 per shift for the hoistman who also sharpened steel, $6 for the lead shaftman, and $5 for his helper. The contractor's costs were: Labor, 2 hoistmen $600; 4 shaftmen $1,320; labor sawing cribbing $40; explosives $410; liability insurance $104; a total of $2,474. The contractor was paid $2,980 for the completed shaft, which was 260 feet deep; 100 feet of the shaft was in shale.

The shaft was on an operating property so that air could be furnished by piping from the mine compressor. The sinking derrick was built to the dimensions of the standard hoisting derrick so that it could readily be converted into a standard derrick. The shaft was sunk dry. The following were the costs of the shaft to the company:

16-foot sinking derrick	$ 160.00
12 by 14 foot "dog" house	45.00
Rent on 30-hp. boiler	60.00
Rent on steam hoist	60.00
Compressed air (estimated)	235.00
Fuel	90.00
Water for boiler	60.00
Cribbing and nails	443.00
Drill and equipment	335.00
Hoisting	110.00
Shaft and blacksmith tools	47.00
Oil	27.00
Sail and blower	144.00
1,600 feet of 2-inch pipe	346.00
Labor	80.00
	$2242.00
Shaft contract price	2980.00
Total cost	$5222.00
Total cost per foot	$ 20.08

The shaft is 5 by 7 feet in cross section and fourteen 5-foot holes constitutes a shaft round; a four-hole diamond cut is employed. The round is shown in Figure 2. The cut holes are drilled 5-1/2 feet in depth and all other holes are 5 feet deep. The side holes are drilled with a slight pitch towards each other. An average of 4 feet of shaft is broken per round, for which slightly more than a box of 30 per cent gelatin powder is used.

Most of the holes in the shale can be drilled with augers, but in the rock all holes have to be drilled with jackhammers. Four cut holes and four corner holes, using 40 sticks of 30 per cent gelatin powder, will advance the shaft 4-1/2 feet per round in shale. It is customary to break three rounds in the shale before cribbing. Where the flow of surface water is not strong, staggered cribbing is used, otherwise it is necessary to use solid cribbing. Care must be taken to put in enough lagging behind the cribbing, because the shale tends to slack upon exposure and thus to put an undue strain upon the cribbing. Figure 3 shows the method of supporting shafts in heavy ground. The foregoing information has been taken from a paper by S. S. Clarke[5], which was read before the Joplin-Miami Section of the American Institute of Mining and Metallurgical Engineers.

When water is encountered in sinking, pump seats are cut in the shaft at convenient intervals, and often as many as five or six of these seats are necessary. Sinking pumps suspended from chains or cable are seldom used in the district.

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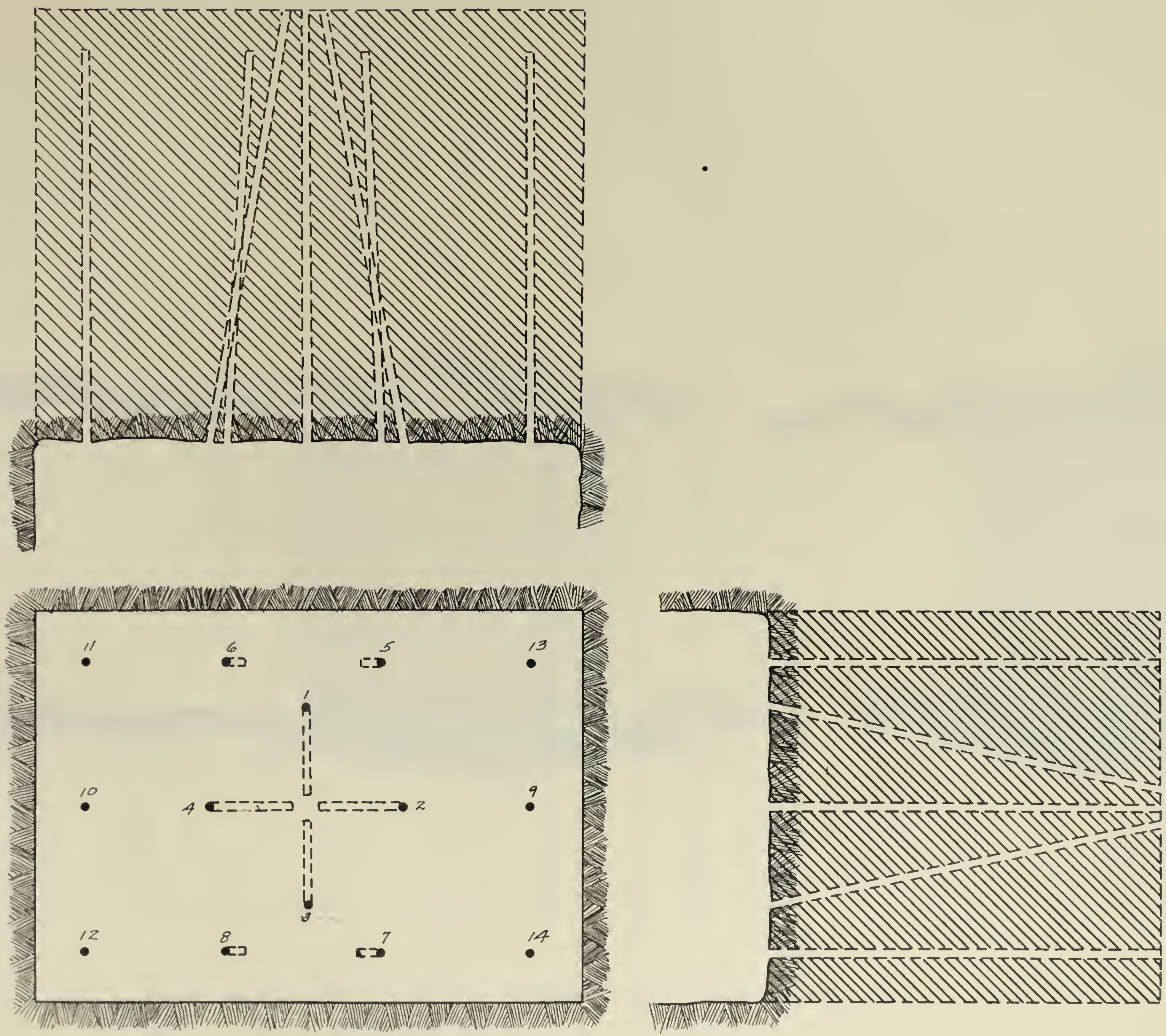
5 Clarke, S. S., "Shaft-sinking in the Tri-State district": Min. and Met., vol. 9, August, 1928, pp. 358-359.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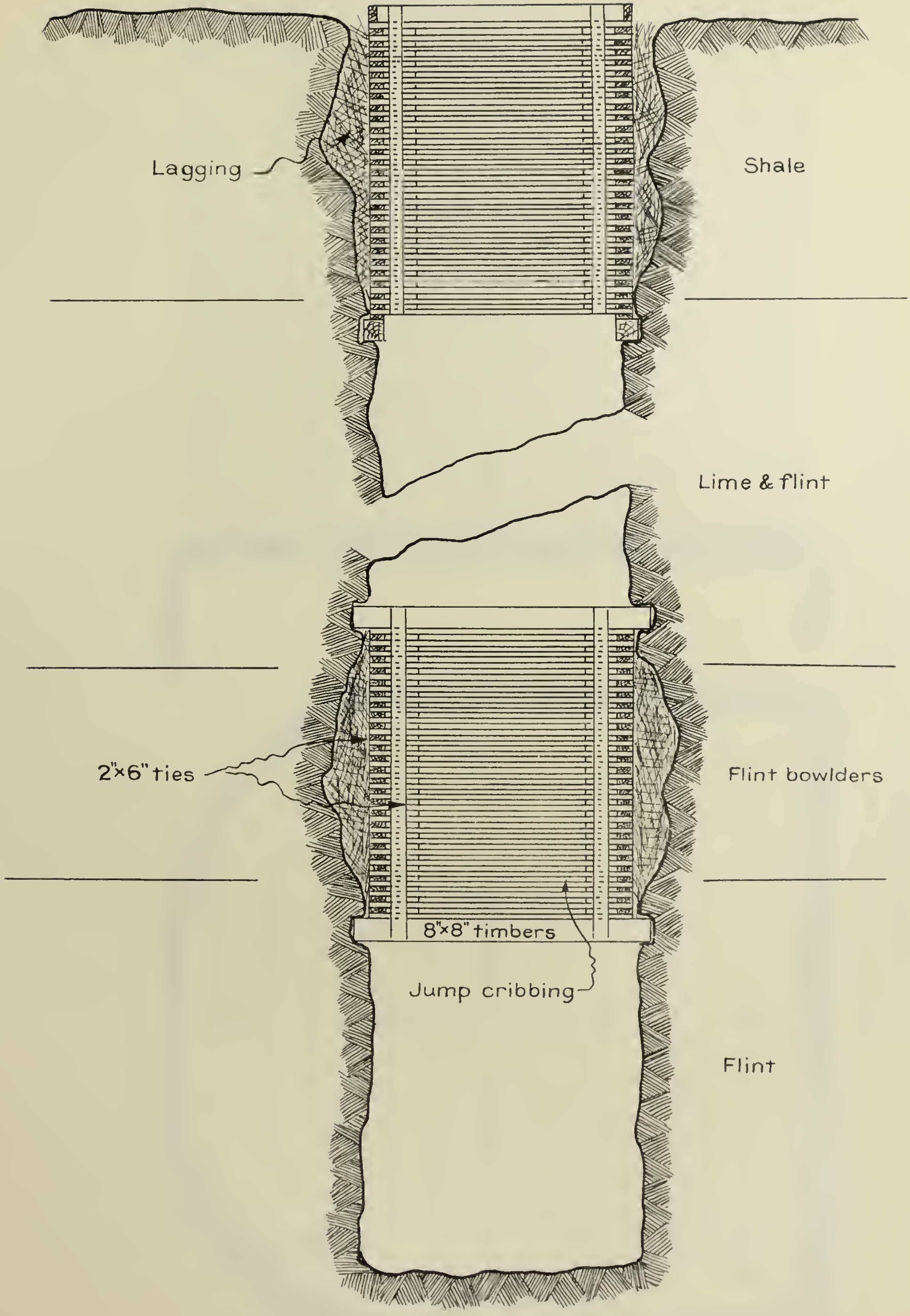

FIGURE 3.—Section of shaft, showing method of support in heavy ground

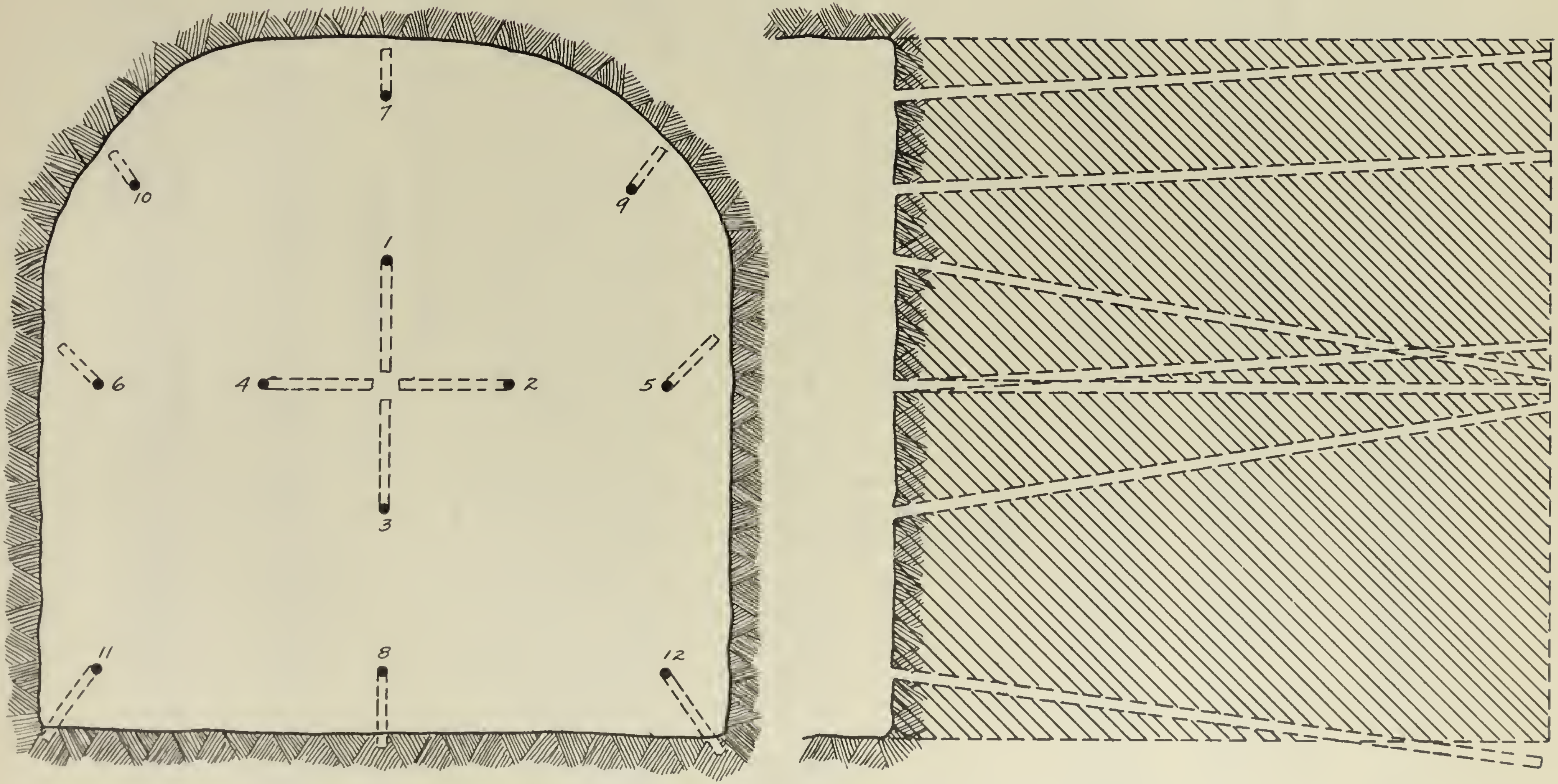

FIGURE 4.—Twelve-hole 8' pull drift round for soft, spongy ground. For ordinary ground two cut holes are omitted

An underground shaft or winze has been sunk from the main to the lower level through loose bowlder ground. It had to be cribbed and lagged very carefully. This shaft is the only connection between the main and the lower levels, and all ore, supplies, and men are handled through it.

"Pull" Drifts.- Prospect of "pull" drifts are driven 7 by 7 feet in cross section. Several drifts were contracted for at the rate of $9 per foot; the contractor furnished all labor, including the hoistman and the powder, and the company furnished the power and all equipment. No total costs are available for these drifts.

A drift was driven on company time at a cost of $6.50 per foot for labor and powder. The machineman was paid a bonus of 25 cents per shift, and the mucking was contracted for at the rate of $7 per round. The costs were as follows:

Drill labor	$ 1.46
Mucking labor	1.17
Hoisting	.79
Miscellaneous labor	.47
Explosives	2.61
Cost per foot	$ 6.50

A 10-hole round with two cut holes, two side holes, three roof holes, and three stope or bottom holes was used. About 24 sticks of 30 per cent gelatin powder were used in each hole, or about 2-1/4 boxes to the round. Usually 6 feet were broken per round.

For soft spongy ground a 12-hole round is used, as shown in Figure 4. The four-hole diamond cut, two side holes, three roof holes, and three stope or bottom holes constitute the round. The holes are drilled 8 feet deep; 7 feet are broken per round, using 2 boxes of 30 per cent gelatin powder. This ground is a soft, spongy, leached lime and flint and is known locally as "cottonrock." The drift is advanced whenever a machine is available from the regular mining operations, and the mucking is done on company time; no costs are available.

Raises.- No raises have been driven in this mine, for no mineable ore bodies have been found above the main level.

PRESENT MINING METHOD

The plan and sections of Mine No. 2 are shown in Figures 1, 5, and 5a. The headings are kept well in advance of the main stope except where the face is low, when no stope is carried; the entire face is advanced by the same method as the heading.

The ore breaks into large bowlders making it necessary to do a considerable amount of "bowlder popping" before loading into cans. All bowlders are drilled with jackhammers before blasting.

For all drilling except the "bowlder popping" the heavy Leyner drill is used. The headings are drilled from a post, but the stopes are drilled from a tripod. For ordinary work 33 per cent ammonia powder is used, but for primers and "bowlder pops," 30 per cent gelatin powder is used. The cost of ammonia powder averaged $0.1315 per pound, and the gelatin powder $0.1300 per pound for the year 1928. The ratio of ammonia to gelatin is 5 to 1.

Underground Support.- The mine is supported almost entirely by pillars of ore from 15 to 70 feet in diameter, with an average of 30 feet. The larger pillars are left in the lean ore zone which runs through the center of the ore body, and the smaller pillars are left in the richer portions of the two ore runs. The pillars are spaced 40 to 60 feet, center to center, with an average of 50 feet.

One section of the mine has a shale roof that becomes dangerous, requiring that the drift be cut down to a "pull" drift size and necessitating the use of several sets of caps and posts with lagging. This method of support is of minor importance. No mud sills are required, for the floor is hard.

The roof stands well if it has been properly arched and carefully trimmed as the faces are advanced. All mines in the district employ a "roof trimmer" whose sole duty is to examine all working faces every day and keep the walls and roof free from loose rock. Most of the trimming can be done from the top of the broken ore or from the heading floor, but for work too high to be reached in this manner the "trimmer" employs a ladder. Ladders are in 20-foot sections, and in very high headings three sections of ladder are often needed. The "roof trimmer's" tools comprise a short bar of 7/8-inch hexagon steel which has been pointed at one end and shaped like a nail bar at the other end. This bar is 6 to 10 feet long. For places that can not be reached with this, a longer bar, made usually of 1-inch pipe, is employed. The "roof trimmer" wears the ordinary carbide cap lamp, but usually a larger reflector replaces the one which comes with the lamp. The reflector from a Model T Ford car with the nickle plating removed makes an ideal lamp reflector, although it is a little heavy for continuous wear.

DRILLING AND BLASTING PRACTICE

Compressors.- Air is furnished by two belt-driven, two-stage compressors. The main compressor has a capacity of 1,050 cubic feet per minute, and the booster has a capacity of 950 cubic feet per minute. The pressure is maintained at 110 pounds at the compressor, which gives an effective pressure of 95 pounds at the machines.

Drills.- Several makes of drills are used but they are all of the heavy Leyner type. Some of the lighter drills can be run as one-man machines, but they are all used as two-man machines at this property. The drills are mounted on posts for heading and "pull" drift work and on tripods for drilling the stopes. All Leyner drills use 1 1/4-inch hollow round steel and the jackhammers use 1-inch hexagon hollow steel.

The standard cross bit with an 18° taper is used. The gauges are dropped 1/4 inch for steel length changes up to 11 feet and then the gauges are dropped 1/8 inch for steel changes up to 20 feet. The starter bits are three feet long with a gauge of 2-3/4 inches. The steel lengths increase 2 feet for each change except the last one, which is 3 feet. The 20-foot steels have a gauge of 1-1/2 inches. The steel consumption is 0.008 pounds per ton of ore mined.

The blacksmith shop is located on the surface near the collar of the material shaft. All drill steels are bitted and shanked here on a power sharpener.

Blasting.- The blasting practice at Mine No. 2 is essentially the same as that used at Mine No. 1, but the character of the ground is different enough to warrant a different type of heading round and different loading of the "squib shots."

The heading rounds are drilled from a 7 1/2-foot post using 10-foot steels. A round consists of seven to nine holes. The nine-hole round is shown in Figure 6, which also shows the manner of drilling the stopes or benches. Two boxes of powder are used for each round, and about 10 feet are broken per round.

The faces are not so high in Mine No. 2 as at Mine No. 1. The high faces average only about 25 feet, so that most stope and splitter holes are drilled only 12 to 15 feet deep. The slope of the stope from the crest to the floor is maintained at about 45°. For a 25-foot face the collars of the splitter holes are about 10 feet behind the crest of the stope, and the collars of the stope holes are the same distance behind the collars of the splitter holes.

The splitter holes are three in number unless the distance between pillars has to be cut down because of unsafe ground, when the number is two. These are drilled in the same horizontal plane and have a rise of 1 inch to the foot so that they can be readily washed free of cuttings after squibbing.

The stope holes are always the same in number as the splitters and are placed practically vertically under the splitters. These holes are drilled downward with the same fall as the rise of the splitters. In order to prevent high bottom they are so located that the bottom of the hole will be below the grade of the drift.

Each splitter and stope hole is squibbed or chambered twice before the breaking charge is loaded. Five sticks of powder are used the first time. The hole is then thoroughly washed, and 24 hours later it is shot with 15 sticks of powder. The hole is now ready for the breaking charge. The breaking charge takes from one and a half to two boxes of 33 per cent ammonia powder for the stope holes and one to one and a half boxes for the splitter holes. The pockets are always loaded to capacity, and two primers are used to insure against a misfire.

All primers are made up with 30 per cent gelatin powder, using a No. 6 cap. For the rest of the blasting, except in the "pull" drifts, 33 per cent ammonia powder is used. The cap is placed in the center of the stick of powder and the entire primer is placed in a safety tube provided for the purpose. Waterproofed fuse with a speed of 2 feet per minute is employed. All charges are stemmed, using clay cartridges made from clay found on one of the company's properties. The clay cartridges cost about $0.01 each.

The superintendent endeavors to maintain his tonnage at 100 cans or about 60 tons per machine shift. This includes machines on heading and stope work. Many of the faces are low, the average height of face in this mine being 14 feet.

LOADING AND TRAMMING

All loading is by hand shovelling into cans set on low trucks. The cans are 32 inches in diameter by 30 inches deep, and have a capacity of 0.6 tons of ore. The track gauge is 18 inches.

All tramming from the "lay-bys" near the working faces to the hoisting shaft is done by mules. There are four mules in the mine, three in daily use and one in reserve for emergencies. On an even grade a mule can pull 12 loaded cans. Figure 1 shows the haulage plan and lengths of hauls.

The northwest part of the mine is 10 feet lower than the main level, and hoists are necessary to haul the loaded cans up the inclines. On the main haulageway a geared, belt-driven hoist is used (fig. 5). A 10-hp. motor furnishes the power. This hoist is capable of handling six loaded cans per trip. Two short inclines from the working faces are operated by small air hoists or "tuggers." These hoists can handle only one loaded can at a time.

The ore from the lower level is brought to the main level by a 52-hp. geared electric hoist.

PUMPING

Ordinarily no pumping is now necessary, since adjoining mines working at lower levels keep the water level below the present workings. In the spring of the year when the rains cause a heavy flow of surface water, the pump at the deep pump shaft is used.

During the summer months the pumps are run to supply mill water for several of the company's mines, which are connected with the pump shaft by a 10-inch pipe line.

The pumps are electrically driven and have a capacity of 450 gallons per minute.

LABOR EFFICIENCY, 1928

Record of labor performed at Mine No. 2 for nine months of 1928:

	Total Shifts	Tons per Shift
Trammers	1,032	84.84
Drill runners and helpers	2,924	29.94
Muckers	3,784	23.14
Miscellaneous Labor	2,064	42.42
Total underground operations	9,804	8.93

Ore hoisted during 1928 amounted to 87,563 tons.

PERCENTAGE OF EXTRACTION

At present 25 per cent of the area mined is left in pillars. It is expected that some of these pillars can be recovered, so that the total extraction will be about 80 per cent. Many of the pillars will be left because they are too lean to pay the recovery cost, but the greater number must be left to protect the surface because of the character of the ground.

WAGE AND CONTRACT SYSTEM.

All labor, except muching, is based on an eight-hour day. Muckers are paid on contract, 11 cents per can of 0.6 tons on the main level and 14 cents per can on the lower level. The bonus is paid to men employed on the lower level because the ventilation is poorer and they can not compete on an even basis with the men on the main level. The following wage scale was in effect during 1928 when zinc ore prices stayed at $40 or under for prime western ore.

Machine runners	$ 4.25
Machine helpers	3.75
Trammers and drivers	3.50
Hoistmen	4.75
Powdermen	4.50
Roof trimmers	4.00
Screenmen	3.50

This wage scale is based on $40 zinc ore. If the price goes to $45 and stays there for one week, all wages are automatically raised 25 cents per shift, and muckers are raised 1/2 cent per can. In the same manner, for every $5 raise in the price of zinc ore above $45 the wages are raised at the same rate, but for every drop of $5 the wages are reduced. Wages are not reduced when the ore price drops below $40, for this is the base price for the wage scale. A week is always allowed between the wage changes to make sure that the market will not fluctuate above or below the critical price.

An average experienced mucker will load 40 or more cans in an eight-hour shift. There are several men at the mine who will average 100 cans per shift, but they are exceptional men and are usually given the best working places.

VENTILATION

The ventilation is natural, except in one place where an ore body is being opened up at the end of a long "pull" drift. A blower has been installed on the surface and air is forced down to this area through a churn-drill hole.

A large blower has been installed at the man shaft to change the draft in the winter time in order to prevent the formation of ice.

MINING COSTS

Below is the total cost of delivering a ton of ore into the mill hopper at Mine No. 2:

Cost of delivering 1 ton of ore into the mill hopper

	Mining	General underground	Surface expense	Total
Labor	0.392	0.084	0.055	0.531
Supervision	0.045	- -	- -	0.045
Compressed air, etc.	0.114	- -	- -	0.114
Power cost	- -	0.025	0.051	0.076
Explosives	0.157	- -	- -	0.157
Other supplies	- -	0.056	0.014	0.070
Total	0.708	0.165	0.120	0.993

The surface expense, which is directly applicable to underground operations, includes the hoist and screen room and the surface tram from the field shafts to the mill hopper; the incline tram into the mill hopper is also included. Ore was mined from field shafts during only a small part of the year; the major part of the ore was hoisted at the mill shaft and dumped directly into the mill hopper.

SUMMARY OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Name or number of mine: No. 2, Tri-State district   Year: 1928, 9 months' operation

Tons of ore mined and hoisted 87,563

Mining method: Open stope with pillar support.

(A) Labor (Man hours per ton)

Breaking (drilling and blasting)	0.297
Mucking	0.345
Haulage and hoisting	0.125
Supervising	0.031
General	0.078
Total underground labor	0.876
Average tons per man-shift underground	8.93
Labor, percentage of total cost	58.00

(B) Power and Supplies

Explosives, 33 per cent ammonia (lbs. per ton)	1.265
Total power (kw. hrs. per ton)	5.75
(1) Air compression	2.51
(2) Hoisting	2.42
(3) Pumping	0.82
Other supplies in percentage of total supplies and power	16.78
Supplies and power, percentage of total cost	42.00

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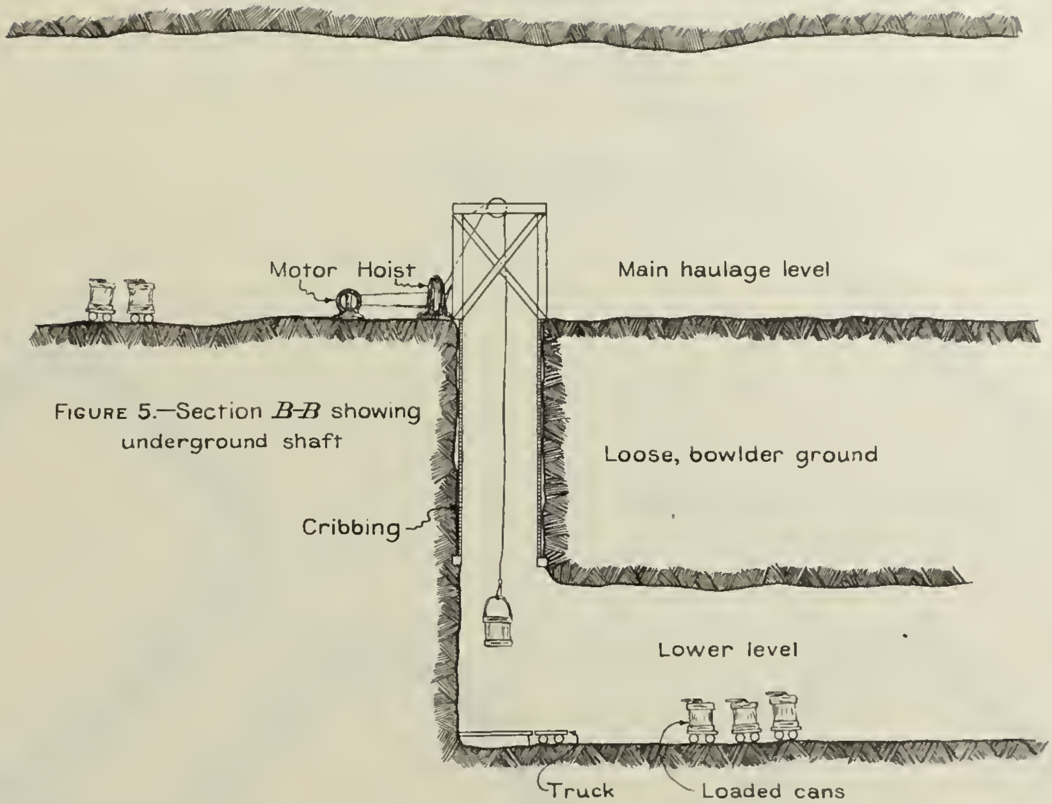

Figure 5.—Section *B-B* showing underground shaft

Figure 5A.—Section *A-A* showing incline-hoist layout

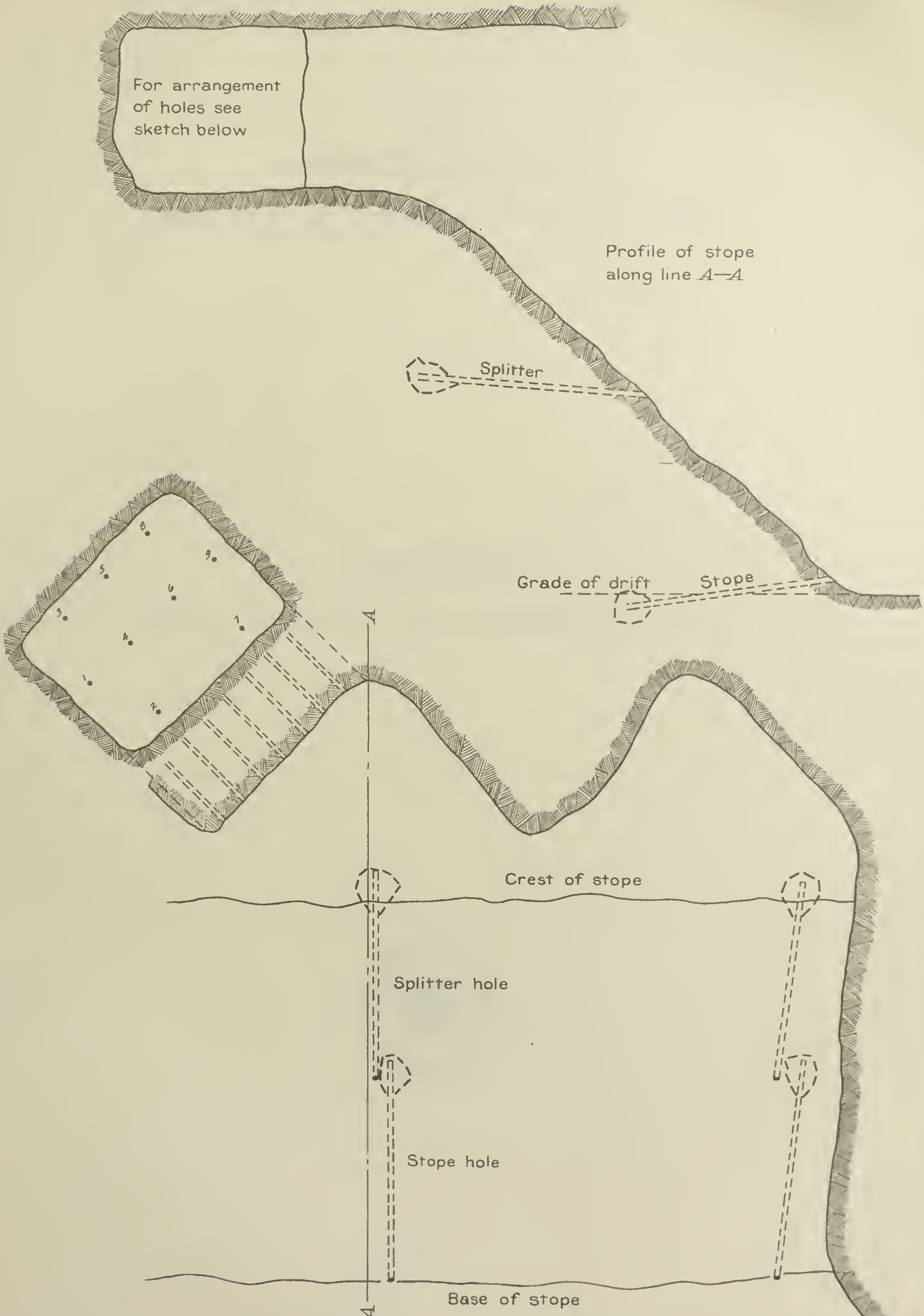

FIGURE 6.—Plan of stope and heading, showing arrangement of holes

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DEPARTMENT OF COMMERCE -- BUREAU OF MINES

GRAPHITE

PART II -DOMESTIC AND FOREIGN DEPOSITS

BY

PAUL M. TYLER

Circular No. 6122.
May, 1929.

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

GRAPHITE.

PART II. DOMESTIC AND FOREIGN DEPOSITS[1]

By Paul M. Tyler[2]

TABLE OF CONTENTS

1 - The Bureau of Mines will welcome reprinting of this article but requests that the following footnote acknowledgment be used: "Printed by premission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"
2 - Assistant to chief, economics branch, U. S. Bureau of Mines.

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FOREWORD

Graphite occurs in many places in the United States, but previous to 1914 the domestic production amounted to only 15 to 20 per cent of the natural graphite consumed in this country. Including artificial graphite, the proportion was about 25 per cent. During the World War the production of both natural and artificial graphite increased until in 1918 it contributed more than one-half of the available supply, and since 1919 the proportion has not fallen below 40 per cent. Except during the war period, however, this apparently increasing independence has not been due to any large increase in production from domestic mines. It simply reflects a larger output of artificial graphite and reduced imports of Ceylon lump and chip, grades which have never been produced in substantial quantity in the United States.

DOMESTIC DEPOSITS[3]

American graphite deposits, though numerous and often large, are characteristically low-grade. They constitute an abundant source of potential supply, but in normal times they have proved relatively costly to work, and even at the same price the various products have never been able wholly to displace certain imported qualities, notably crucible grades from Ceylon and pencil graphite from Mexico.

Marked changes in the nature of consumption have reduced the demand for crucible grades of graphite, so that differences in quality between the domestic and foreign supplies tend now to be of less significance than formerly. These changes likewise have increased the proportion of the total consumption that can be satisfied by artificial graphite whose range of usefulness includes many of the applications of crystalline as well as of amorphous grades. Since there is normally a surplus of petroleum coke and an ample supply of anthracite coal, the production of artificial graphite could doubtless be expanded readily if necessary -- probably much faster than the output of crystalline graphite, all of which has to be concentrated. In Alabama and elsewhere, however, there are many idle mills that could probably be started again, rather promptly. The principal known deposits of crystalline flake graphite in the United States are in Alabama, California, New Jersey, New York, Pennsylvania, Texas, and Virginia. Other localities include Alaska, Colorado, and North Carolina. In Montana is the only known domestic deposit of vein graphite. Amorphous graphite is found more or less abundantly in Colorado, Michigan, Nevada, Rhode Island, New Mexico, and Wisconsin, and there are great beds of graphitic schists of possible commercial significance in Maine and in South Dakota, Wyoming, and certain other western States.

3 - The data herein given with respect to domestic deposits have been taken mainly from Mineral Resources chapters prepared by Bastin, Ferguson, and Middleton, or from other published sources, and are supplemented by a very limited amount of field work by the writer.

Alabama[4] - Moderate-sized flake graphite, rather smaller than that produced in certain other localities, occurs in Alabama in bands of graphitic schist in Clay, Coosa, and Chilton Counties in the vicinities of Ashland, Mountain Creek, and Good Water, respectively. The schist usually dips steeply to the southeast and, associated with granitic intrusions, occurs in lenses up to 100 feet wide in metamorphic rocks, usually pre-Cambrian schists similar to the ore itself but containing more mica. According to Dr. W. B. Jones, Alabama State Geologist, graphite outcroppings are found in four counties in 16 different beds, of which 13 are of commercial importance. He considers the supply limitless.[5] The rock is largely quartz with some mica and apatite, and with rarely more than 3 or 4 per cent of graphite. The average war-time yield per ton mined was about 19 pounds of No. 1 flake, 5 pounds of No. 2 flake, and 3 pounds of dust -- a recovery of a trifle more than 50 per cent of the total graphite, which averaged about 2½ per cent in the rock as mined. Subsequently, more efficient milling has made possible a 75 per cent recovery from rock containing only 2 per cent of graphite, the yield per ton being about 14 pounds of No. 1, 4 pounds of No. 2, and 7 pounds of dust (mainly No. 3 dust containing 75 to 85 per cent carbon). Usually a small amount of overburden (1 to 6 feet) has to be removed, and mining is stopped at 30 to 60 feet where hard, unweathered "blue rock" is reached. The deposits are worked opencast, and few mines are as much as 75 feet deep. The weathered ore can be mined largely by hand picks and is much easier to mill than the hard ore. The latter, moreover, contains a small amount of pyrite, which is not present in the weathered horizons of the ore bodies. Amorphous graphite is also found in Alabama in the feebly crystalline schists or the Talladega slates,[6] as a black graphitic clay free from grit. Occurrences of this sort have been observed near Millerville in Clay County and in the vicinity of Blue Hill in Tallapoosa County, but such material is difficult to separate and has not yet been worked commercially to any great extent.

Graphite mining in Alabama was attempted as early as 1888 and was established in a small way in 1900. The chief development of graphite in Alabama, however, occurred during 1917 and 1918, when for a time there were 39 concentrating mills in the area, over one-half of which were in actual operation. Activity continued after the Armistice until later in 1920, when the industry collapsed, remaining almost at a standstill until 1926. In 1927 three operators reported sales of 3,474,000 pounds of crystalline graphite, or two-thirds of the total for the United States. Though only a little less than one-half the record sales made in 1918, the 1927 output was more than double the annual production in pre-war years.

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4 - Prouty, W. F., "Geology and mineral resources of Clay County": Rept. No. 1, Geol. Survey of Alabama, 1923. Brown, John S., "Graphite deposits of Ashland, Alabama": Econ. Geol., vol. 20, No. 3, May, 1925.

5 - Statement before Ways and Means Committee, Tariff Readjustment Hearings, 1929.

6 - McCalley, H., Alabama Geol. Survey Rept. Valley Regions, pt. 2, 1897, pp. 36-38.

In 1927 the producing companies were the Bama Graphite Mines, successor to the Flaketown Graphite Co., The Superior Flake Graphite Co., and the Southwestern Consolidated Graphite Co. which has been operating for a number of years in Texas and recently took over and operated the plant of the Ceylon Co. at Hollins, Ala. The Alabama-Quenelda Graphite Co., successor to the Quenelda Graphite Corporation, expects to operate during 1928. According to Bastin,[7] the quarry and mill of the Quenelda Co. were located a little over eight miles west of Ashland, to which the concentrate is hauled for shipment. The rock mined is highly schistose, and is composed largely of quartz and graphite. Feldspar and mica are rare, but a white fibrous mineral (probably sillimanite) is abundant. The original mill which combined wet and dry concentration was destroyed by fire. The property of the Flaketown Graphite Co. is in Chilton County, about 3½ miles northeast of Mountain Creek (the nearest shipping point). The rock is a graphitic quartz schist of the same general type as those mined in the vicinity of Ashland. Small quantities of green micaceous mineral, probably muscovite, are present in some specimens, but in general mica is rare. A dike of granite pegmatite 1 foot wide intrudes the graphitic schist at the main pit, lying parallel to the foliation. Within 1 or 2 inches of the schist it carries graphite in scattered flakes up to 1/8-inch in diameter. An analysis made by the United States Geological Survey of a composite sample of graphitic schist collected from a number of different exposures of this property, showed 4.63 per cent of graphite. During part of the year electric power for the mill is generated by water power from a 24-foot dam on Chestnut Creek. Auxiliary steam power has also been installed.

Alaska - Crystalline graphite was produced intermittently in Alaska from 1905 until 1918. Most of the output was in the nature of trial shipments, chiefly to experimental mills in the State of Washington, although a little of it was manufactured into foundry facings in San Francisco. The deposits which occur on the northern and southern slopes of the Kigluaik Mountains, in the southern part of Seward Peninsula, have been visited by geologists of the United States Geological Survey. The lenses of graphite occur in association with quartz schists carrying biotite, and garnet schists carrying some calcite. Granitic rocks are found nearby. Locally there are two or three series of graphite lenses which are parallel in strike and dip. It is not known whether these are distinct horizons, or merely result from faulting or folding. There are a number of exposures showing lenses varying in thickness from one foot to several feet, and it is reported that these deposits of graphite offer an opportunity for large production. Transportation problems are simple, as the ore could be carried to Graphite Bay, a good shallow-water harbor.[8]

Arkansas - In Arkansas graphite has been reported in the western part of Pulaski County and also in Crawford County. Some development was in progress in Crawford County in 1910 on material said to run 40 to 60 per cent in carbon, but Bastin, who examined the prospects in 1912, reports that the deposit was

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7 - U. S. Geological Survey, Mineral Resources of the United States, 1913: Pt. 2, pp. 191-193.

8 - U. S. Bureau of Mines, Mineral Resources of the United States, 1917: Pt. 2, pp. 113-115.

not graphite but was a thin bed of carbonaceous shale.

California - California resumed production in 1927 after a lapse of four years. The Standard Graphite Corporation revived a property which, though worked intermittently for 30 years, seemed incapable of commercial production before the flotation process was introduced. The ore is crystalline graphite occurring in small flakes finely disseminated through schists which are locally enriched up to a thickness of 100 feet, and contain lenses of high-grade ore varying in width from 6 inches to 15 feet. Mining is conducted by means of shallow tunnels and open-cuts.[9]

This mine is situated about 15 miles north of Los Angeles, and is characteristic of other deposits in Los Angeles and San Diego Counties, all of which resemble in a general way characteristic occurrences in New York, Pennsylvania, and Alabama. The California flake, however, is much smaller than that found in many of the eastern deposits, often being only 0.25 millimeter in diameter. The percentage of graphite, however, is higher, and due to the fact that mica is practically absent, it is fairly easy to concentrate by modern methods into a product suitable for foundry facings and paints. Probably products even purer could be made if necessary, but the smallness of the flake makes it of little value for crucibles or refractories, although it is probably suitable for lubricating stock. Before the World War some graphite was mined in Calaveras and Mendocino Counties.

Colorado - Amorphous graphite was produced fairly steadily in Colorado before 1922. A considerable portion of the output was sent to a mill at Warren, Ohio, where it was prepared for use in paints, foundry facings, etc. The deposits appear to be related in origin with the Cretaceous coal deposits of the Crested Butte area. Anthracite coal has also been found nearby. In addition to the main bed, there are a number of subordinate beds, interbedded with white to gray crystalline limestone, buff quartzite, and gray to purple silicious schists. A dike of fine-grained granite cuts the sediments a few feet from the main graphite bed, which varies in thickness from 3 to 4 feet, and about one-half of which has been second-grade ore containing more clay and less graphite than the first-grade ore, which is dull black and very pure. The first-grade graphite used to be packed in bags, while second-grade was shipped in bulk. The property of the Federal Graphite Co., the principal producer, was situated northeast of Turret in Chaffee County, 5 miles from a siding of the Denver & Rio Grande Railroad. Since 1916 graphite has been produced in some quantity in Gunnison County, near Pitkin, about 10 miles from the railroad.

Georgia - Georgia has produced no true graphite, but a large quantity of slate containing 2 to 15 per cent carbon quarried near Cartersville in Bartow County has been ground for use as a filler and dryer in fertilizers. For some years prior to 1910, the output of this material was included in the production statistics, but since it is not adapted for any of the purposes for which higher grades of amorphous graphite are used and since these higher grades are not used

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9 - Hubler, W. G., "Concentrating graphite in southern California": Eng. and Min. Jour., vol. 125, June 30, 1928, p. 1059.

as fertilizer filler, this material is no longer classed as graphite. It formerly sold at $1.25 to $1.50 a short ton f.o.b. mills. The deposits have been described by Hayes and Phalen.[10]

Idaho - In 1908 a graphite mine was opened in Ketchum, Blaine County, Idaho, and a little amorphous graphite was reported to have been produced in the following year. Graphitic schists occur near Salmon River, near Grangeville; a specimen analyzed by the United States Geological Survey in 1912 contained 7.67% fixed carbon.

Maine - Graphite is known to occur in several localities in Maine, and in 1905 an unsuccessful attempt was made to mine it near Madrid in Franklin County. According to George Otis Smith[11] graphite occurs at Madrid in very small particles, ranging from 0.5 to 0.20 millimeters in diameter, fairly evenly distributed in a schist near its contact with granite pegmatite. A sample analyzed 8.5 per cent graphite, but it would be extremely difficult to prepare a marketable product because the graphite, in addition to being so small, is intimately associated with mica flakes (muscovite). In Cumberland County, 1½ miles northwest of Yarmouth, graphite occurs as disseminated flake in a dike of granite pegmatite. The sample analyzed 9 per cent graphite by weight, but as the dike is only 1 foot wide, the deposit is not considered to have any immediate value.

Massachusetts - Considerable historical interest attaches to a graphite deposit near Sturbridge between Worcester and Springfield in Massachusetts. This mine was not only the first graphite mine worked in the United States, but was one of the first mining ventures of any kind in America. It was first secured by a grant to John Winthrop, jr. (son of Governor Winthrop of Massachusetts), who also purchased the tract from the Indian inhabitants between 1644 and 1658. The preliminary operation appears not to have been successful, but in 1738 mining was resumed, and one or more shipments were made to England, where the material brought about 4 pence a pound. Less than 1 ton was extracted in the summer of 1740, and there is no record of further development until 1828-29, when it was operated by Frederic Tudor of Boston as an adjunct to the manufacture of crucibles.[12] After again being idle for many decades it was eventually sold in 1902, after which it was operated intermittently for a few years. One lump of solid graphite weighing 510 pounds was reported to have been taken out in 1904.

The graphite from this mine apparently was similar to the occurrences in Ceylon and at Dillon, Mont., but apparently it formed a series of disconnected lenses or flat pockets, mostly less than 3 or 4 inches thick, conformable with the bedding but not very persistent. The containing rock was quartzose mica schist intruded by small masses of granitic rock. The main lode has been prospected to a depth of 50 to 60 feet for a half mile or more, and the chances of further production are not favorable.

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10 - Hayes, G. H., and Phalen, W. C., Graphite deposits near Cartersville, Georgia: U. S. Geol. Survey Bull. 340, 1908, pp. 463-5.

11 - Smith, G. O., Graphite in Maine: U. S. Geol. Survey Bull. 283, 1906, pp. 480-483.

12 - Hayes,, G.H., "The tale of Tantiusques, an early mining venture in Massachusetts": Am. Anti. Soc. Proc., Ann. Mtg., Oct. 31, 1901. (Reviewed in Mineral Resources, 1913.)

Michigan - Graphitic slate, classed as amorphous graphite, has been produced in considerable quantities near L'Anse in Baraga County, Mich. The rock mined is a dark reddish-brown graphitic and ferruginous material said to average 33 to 35 per cent of graphitic carbon. After careful sorting it is crushed, ground, air-floated, and bolted through 200-mesh silk. It has been used for paint pigment, and has been employed in the manufacture not only of black paints but, mixed with other pigments, also of gray, dark-green, and dark-red paints.

Montana - The only graphite property operated in Montana is that of the Crystal Graphite Co., near Dillon. This deposit yields vein graphite very similar to Ceylon material. The rocks immediately associated with the graphite deposits are (Paleozoic?) limestones and quartzites, which lie above schists and slates that are probably pre-Cambrian. The sedimentary formations are intruded by granite and pegmatite. There are several graphite veins, the main vein showing from 2 to 8 inches of very pure graphite. In one of the exposures the vein is 2 feet wide and consists of an irregular network of graphite veinlets enclosing the many shattered fragments of schist graphite forming about one-half the vein material. According to Bastin,[13] the graphite of this mine is in the form of rather irregular veins deposited along fracture planes and more or less broken by later fracturing. Particularly near the surface, thin films of iron oxide often occur along parting planes between the graphite plates. The graphite is a little softer than that from Ceylon and grinds up more easily, but its lustre is not quite so brilliant. These differences may be due to the fact that the workings are still confined to rather shallow depths. Truly fibrous varieties such as the needle lump, which occasionally occurs in Ceylon, have also been noted in Dillon. Substantial quantities of graphite were produced from this mine during the war period and shipped to eastern consuming centers, but since 1918 the output has been very irregular. In 1927 the company was active mainly in perfecting a milling process.

Deposits of flake graphite in Montana were prospected in 1918, but the profitable operation of such deposits so far distant from consuming centers will prove difficult.

Nevada - The Carson Black Lead Co. has produced amorphous graphite quite steadily for more than 20 years from its property in Ormsby County, 3 miles from Carson City. The material, a black graphitic shale, is said to carry 30 to 50 per cent of graphite; a composite sample from 20 different places across the main pit when analyzed by the United States Geological Survey showed 49 per cent carbon. Separation of the graphite from its associated minerals is not economically feasible, but the shale after fine grinding is suitable for use as paint pigment and as foundry facings. Most of the production to date has been used for the manufacture of graphite paints in a small mill at Carson City operated periodically by the company. The deposit appears to be moderately large, as the shale is traceable for half a mile along the strike and, in places at least, is 100 feet wide. The graphitization may be due to the intrusion of pegmatite granite, which cuts the shale. Other deposits of amorphous graphite have been

13 - U. S. Geological Survey, Mineral Resources of the United States, 1913: Pt. 2, pp. 202-4.

reported in Nevada, but apparently none of them have been commercially productive. A specimen sent to the United States Geological Survey from Ludwig, Lyon County, was of good quality, and was said to have come from a deposit 4 or 5 feet thick which was visible for several hundred feet along the surface.[14]

New Jersey - Graphite occurs as disseminated flake in the crystalline rocks of the highlands of New Jersey at a number of localities. According to Bayley and Stewart[15] the graphite occurs in several ways: (1) As a component of Franklin limestone; (2) in gneisses which may be in part altered sediments but in places are mashed pegmatites; (3) in coarse granite dikes and pegmatites; and (4) in fine-grained quartzitic micaceous schists, especially where they are associated with pegmatites. The last-named mode of occurrence is the most important. Pegmatites are only graphitized close to the schists, or where they carry fragments of schist.

The latest development is at Annandale where a large quantity of graphitic schist has been opened up. In an open-cut at the bottom of a hill this schist, some of it heavily charged with graphite, has been exposed for a length of 200 feet, roughly, and to a depth of 50 to 75 feet in places. The strata in this vicinity are highly contorted and the workings follow the axis of a well-developed anticline. Tongues of feldspathic rock intrude the schist and in places the schist itself contains bands and small stringers of feldspar, suggesting the presence nearby of pegmatites. Beyond this cut the ground rises abruptly and graphite has been found in drill holes, test pits, and open-cuts at various places beneath the wash (mostly from 3 to 10 feet deep) over an area several thousand feet long and probably a thousand feet wide. A quarter of a mile or more from the lower workings, in the side of what is known as Graphite Hill, a small tonnage of milling ore averaging from 7 to 9 per cent carbon was mined in 1928 by means of a power-shovel, and in this excavation samples of schist found at a depth of 15 feet were said to assay $34\frac{1}{2}$ per cent carbon.

Owing to lack of information as to the thickness of the deposit, it is not possible to make estimates as to the tonnage available, but the development records indicate that possibly millions of tons are within easy reach by surface mining. Moreover, the average grade of the ore seems to be as high/as or higher than that of other known deposits in New Jersey where the graphite content has usually been reported as being between 4 to 8 per cent, with occasional samples showing as much as 11 per cent.

Because of their nearness to consuming markets and their moderately high-carbon content the New Jersey graphite deposits have repeatedly attracted the attention of investors, and numerous attempts have been made at various times to utilize the deposits commercially. Concentrating works were erected at Bloomingdale, High Bridge, and near Brookside; all these ventures failed, however,

14 - Bastin, E. C., Mineral Resources of the United States, 1913: Pt. 2, p. 204.

15 - Bayley, W. S., and Stewart, C. A., "Note on the occurrence of graphite schist in Tuxedo Park, N. Y.": Econ. Geol., vol. 3, 1908, pp. 535-538.

and no further attempts were made from 1907 to 1925. From the description of the deposits it would seem that the earlier difficulties may have been due in part to the intergrowth of biotite mica and possibly of pyrite (both of which are present in some deposits) and in part to the tendency of the flake to be destroyed by the sharp quartz grains in the process of grinding. "Pancaking" also may have been a factor. In 1925 the Annandale Graphite Co., after some preliminary exploratory work on previously known deposits, discovered and began the development of a new ore body situated along the tracks of the Central Railroad Co. of New Jersey, 55 miles from New York and practically on the concrete highway between New York and eastern Pennsylvania. In February, 1927, a large mill using the dry process was completed, but it failed to produce a concentrate containing more than 32 to 38 per cent carbon. After several months of experimental work a section of this mill was remodeled and run for three months as a pilot plant, producing a concentrate said to average day after day practically 97 per cent of graphite. Extraction was reported as over 90 per cent, and the tailings averaged only one-half per cent carbon. The new process involves graded crushing, first in jaw crushers and rolls and then in rod mills with intermediate screens or classifiers at each stage to remove both the clay and the flake graphite as soon as they are liberated from the rock. The overflow from the various classifiers (all minus 65-mesh) is then treated in K-and-K flotation machines, the froth from which is reground in a rod mill before passing to the K-and-K machines used as cleaners. The cleaned concentrate is pumped to a thickener, then to a vacuum filter, and is finally dried ready for shipment. The concentrate as thus prepared may run over 97 per cent graphitic carbon, but a further cleaning process has been devised for increasing the carbon content to over 99 per cent in graphite for lubricating purposes. A product known as "Lubricite" is being put up in packages for sale to automobile owners and preparations are being made for the marketing of various other specialties, utilizing both large and small flake. The tailings from the ore mined in certain parts of the deposit consist of clean sharp sand, sufficiently high in silica to be sold to manufacturers of refractories. In other portions of the deposit, particularly where the rock has not been weathered, the feldspar content is so high that the tailings could be used only for concrete, road building, etc.

New Mexico - Small quantities of amorphous graphite have been mined in the past from a large deposit that occurs in the canyon of the Canadian River about 7 miles southwest of Raton, in Colfax County, N. Mex. The bed, which lies practically horizontal, has been prospected for a distance of several miles along the outcrop and laterally into the principal coal bed of the Raton field which contains bituminous coking coal. In many places where igneous material has been forced into the coal measures, coke has been formed, but in the Canadian canyon the intrusive mass took the form of numerous sills above, below, and in the coal bed, with the result that coal was completely graphitized, especially where the bed was fractured and the diabase was forced into it. The graphite occurs in pockets in the diabase and is more or less columnar. A representative sample taken where the bed was about three feet thick showed fixed carbon, 76.11 per cent; volatile matter, 6.07 per cent; ash, 16.51 per cent; and moisture, 1.31 per cent. In 1889 some 250 tons of this graphite was shipped to Pennsylvania where it was found suitable for the manufacture of paint; the refined product contained 80 per cent of carbon, the remainder being mostly silicon. The mill

was being taken apart for shipment to Raton when it was destroyed by fire. Nothing has since been done toward developing this graphite.[16]

New York - For many years New York produced more graphite than any other State. The deposits, which are situated in the foothills of the Adirondacks, mainly in Essex, Warren, Washington, and Saratoga Counties, were worked before the Civil War. The largest graphite mine in the United States was at Graphite, near Lake George and 11 miles south of Ticonderoga. This mine (which was operated by the American Graphite Co., a subsidiary of the Joseph Dixon Crucible Co. of Jersey City, N. J.) was closed in 1921, and several other mines in the vicinity are reputed to be worked out or abandoned as unworkable; nevertheless, the resources of the State are by no means exhausted.

The ore contains on the average about 5 or 6 per cent graphite. While the carbon content is thus more than double the average for Alabama, the ore is unweathered and hard, resembling in this respect the deeper portions of the Alabama deposits, which portions ("blue rock") are not mined. Two classes of deposits have been observed -- one comprising contact metamorphic deposits in limestone, adjacent to pegmatite, and another occurring as regional metamorphic deposits, not far from the contact of the Hague garnet-sillimanite gneiss with the Faxon limestone or Swede Pond gneiss. The former class, although it contains large flakes of graphite, has proved too erratic and pockety for profitable development. The regional metamorphic deposits on the other hand have proved quite regular and uniform. The ore beds range in thickness from 10 to 25 feet, averaging possibly 15 feet. They dip 25 to 35°, so that as mining progresses, open-pit mining rapidly gives way to underground operations employing ordinarily the room-and-pillar system with underhand stoping. Power drills have been used at most mines, as the rock is hard.

The graphite deposits of New York, including prospects and abandoned mines as well as those which were operated during the war, are described in great detail in a report by H. L. Alling[17] who distinguishes the deposits of the northern area, mainly limestone contact deposits, from those of the southern area which are characteristically bedded. On the map the line runs only a trifle north of Lake George and considerably to the south of Lake Champlain. The northern area includes the deposits of Essex County and sundry prospects and abandoned mines in Ticonderoga Township, none of which have been operated recently, as they have proved too uncertain, too pockety, and too small in extent to be minable under present conditions.

The graphite schists of the southern area, the commercial deposits of the Adirondacks, Alling divides into four groups in order of their relative economic importance, as follows:

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16 - Lee, W. T., Graphite Near Raton, N. Mex.: U. S. Geol. Survey, Bull. 530, 1913, pp. 371-374.

17 - Alling, H. L., The Adirondack Graphite Deposits: New York State Mus. Bull. 199, Albany, 1918.

1. The normal quartz schist, carrying 5 to 7 per cent graphite, which comprises two beds in some properties and one bed in others and has an ore horizon of from 3 to 30 feet thick which tends to be micaceous both above and below. The mica content increases with increase in the feldspar content.

2. A feldspar-quartz schist, carrying 6 per cent graphite and 10 per cent micaceous minerals.

3. Another phase of the quartz schist, containing pyroxene and tourmaline in which the graphite has been redistributed so that certain layers are abnormally rich.

4. A meta-arkose, which consists almost entirely of potash feldspar, a specimen containing only 1.6 per cent graphite.

Only the material of the first group can be considered as graphite ore, but Alling estimates that the properties of Faxon, Hooper Bros., The Flake Graphite Co., and the Graphite Products Corporation, have a combined reserve of 10 to 13 million tons of graphite schist, one-half of which is readily available. Assuming that 3 per cent of the graphite is recoverable, these reserves should yield 3,000,000 pounds of graphite - enough to supply the normal requirements of the United States for this quality of graphite for 50 years.

North Carolina - Graphite is known to occur in many localities in the central and western parts of North Carolina, but attempts to work these deposits have generally failed. The principal deposits of the State have been briefly described by the North Carolina Geological Survey.[18]

A little flake graphite was shipped in 1911 from a deposit 4 miles north of Franklin. Some amorphous graphite was produced in 1916 in Catawba County. Early in 1919 a small quantity of crystalline graphite was mined near Shelby, N. C., by the General Graphite Co. of Birmingham, Ala.; this company also reported development work on South Carolina deposits.

Pennsylvania - Graphite has been mined intermittently in Pennsylvania from an early date. The principal producing deposits are situated about 30 miles west of Philadelphia, and occur in the gneiss of Pickering Valley, Chester County. With one exception the operations have been confined to soft, disintegrated material, analyzing $3\frac{1}{2}$ to $4\frac{1}{2}$ per cent graphitic carbon. The deposits usually dip 35 to 50° and have been mostly worked opencast, although there has been some underground mining employing room-and-pillar methods branching off from shafts or tunnels. The various operations have been described in some detail by F. Bascom.[19] In 1920, the Graphite Products Co., Beyers, Pa., reported a production from its mine at Uwchlan. Since then no production has been

18 - Pratt, J. H., The Mining Industry in North Carolina during 1901: North Carolina Geol. Survey Econ. Paper 6, 1902, pp. 69-72.

19 - U. S. Bureau of Mines, Mineral Resources of the United States, 1919: Pt. 2, pp. 318-322.

reported from the Pennsylvania deposits, which apparently have all been idle.

Rhode Island - Graphitic shales have been worked in Rhode Island for many years. The deposits which occur in the vicinity of Providence and near Tiverton in Newport County represent a phase of the adjacent coal deposits. They were first developed successfully in 1889, but prior to 1898 none of the ventures appears to have been particularly profitable. The principal operation has been at Fenner Ledge, in Cranston, a suburb of Providence. This mine started as an open pit but has been continued underground. The material, which is associated with anthracite coal, is produced cheaply and is sold at a low price chiefly to foundry-facing manufacturers and paint grinders.

South Dakota - Amorphous graphite containing from 30 to 35 per cent carbon and about 40 per cent silica has been found in the Black Hills, notably about 1.4 miles (by wagon road) north of Oreville in Pennington County. Samples of this material seemed suitable for the production of paint pigments or foundry facings, but the distance from consuming centers represents an obstacle which apparently has not yet been surmounted.

Texas - In the central part of Texas, between Llano and Burnet, are situated deposits containing from 6 to 10 per cent graphitic carbon. The yield of No. 1 flake is rather small because amorphous graphite is present and because, as the gangue is hard, much flake is destroyed in grinding. The economic value of these deposits which occur in Packsaddle Mountain, a silicious pre-Cambrian formation intruded by granite pegmatite, is indicated by the fact that one company successfully continued operations after the war during a period when nearly all the other producers of crystalline graphite in the United States were idle. The principal producing company is the Southwestern Consolidated Graphite Co., a Boston concern (which has lately purchased another property, in Alabama). At the property of the Southwestern Graphite Co. the overburden, which varies in thickness from 2 to 8 feet, contains 3 to 4 per cent graphitic carbon. The graphite, however, occurs chiefly in large boulders contained in the hard-baked detritus along with considerable clay and vegetable matter. The ore-body is fully 100 feet thick and a mile long and it dips about 65 degrees southeast. It is mined by steam shovel, and a wood-fired churn drill is used for drilling the holes preliminary to blasting. Five-ton motor trucks with automatic dump bodies are used for haulage to the mill, which now employs K-and-K flotation machines.[20]

Utah - During 1909 a mine was opened in Brigham in Box Elder County, Utah. In that vicinity there appears to be a fairly long belt of carbonaceous schists, forming part of an extensive series of altered sediments. The beds where exposed are 20 feet thick and appear to be persistent along the strike. An analysis made by the United States Geological Survey, however, showed no graphite, although the material contained an average of 3.48 per cent fixed carbon and selected samples showed 5.59 per cent. Apparently the material might be used for paint, although it has not yet been exploited on any substantial scale.

20 - Menardi, H. B., "Modern flotation plant for graphite": Rock Products, vol. 31, No. 18, Sept. 1, 1928, pp. 74-7.

Virginia - Graphite is known to occur in a number of localities in the Piedmont region, east of the Blue Ridge in Virginia.[21] Mention has been made in published reports of large exposure of graphitic schist at Somerset in Orange County (Somers Place) and also of specimens from Louisa County near Green Spring Depot, in Charlotte County on the road from Drakes Branch to Saxe, and in Powhatan County near Jefferson. Many years ago there was a graphite mine about 2 miles west of Buck Mountain in the northern part of Albemarle County. This mine was visited by Bastin in April, 1911, and was described by him in the chapter on graphite in Mineral Resources for 1913. The only visible exposure is on the south wall of an open pit where the rocks are much weathered. Here the pyroxene-syenite is cut by a single vein which varies from 1 to 2 inches across and branches off into the country rock in numerous short veins. The vein is composed largely of graphite which in places is earthy in texture, but in others is crystalline with a well-developed fibrous structure. The description of some hand specimens closely resembles that of Ceylon graphite. In 1915, a new shaft was sunk on this old property with encouraging results. In 1915, also, prospecting for graphite was carried on $3\frac{1}{2}$ miles from Front Royal in Warren County. A sample examined by the United States Geological Survey consisted of schist containing graphite in minute flakes.

Washington - During the World War some development work was done on a graphite property on the east slope of the Cascade Range in Chelan County, several miles west of Nason Creek, Wash. Evidently there were opened up several bodies of schists containing a small percentage of graphite and occasional small veins or lenses up to 18 inches wide of richer material. The graphite particles are mostly under 0.5 millimeters in diameter. Although this graphite might be used for paint pigment, foundry facings, boiler-scale preventives, and possibly stove polish, it would be rather difficult to separate. Cost of milling and high freight constitute serious handicaps to the commercial utilization of these deposits.

Wisconsin - Amorphous graphite has been produced quite regularly in the vicinity of Junction City, Portage County, Wis. The product is crushed, dried, pulverized, and air-floated. In this condition it has been worth at times from \$25 to \$35 per ton. The carbon is said to range from 30 to 40 per cent. In both the principal properties the rock is a black slate or shale. The carbon is in such a very finely divided state that concentration is impracticable. It is uncertain whether the carbon is true graphite or merely amorphous carbon.

Wyoming - According to a bulletin by the State geologist[22] amorphous graphite is found in Laramie, Albany, Carbon, and Fremont Counties, the carbon content, as shown by analyses of samples, varying from 10 to 60 per cent. In Albany County there is a large district known as Plumbago Canyon, where there

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21 - Watson, Thomas L., Mineral Resources of Virginia: 1907, pp. 188-190. Published by Jamestown Exposition Commission.

22 - Jamison, C. E., Mineral Resources of Wyoming: Wyoming State Geol. Bull. 1, ser. B, 1911, p. 34.

are a great many deposits of graphite, and in Fremont County a deposit of graphite, said to be extensive and very pure, occurs near Miners Delight. Deposits in the Haystack Hills, Laramie County, occur in pre-Cambrian mica schist near its contact with granitic intrusive rocks. The largest band of graphitic schist observed has a thickness of only 8 to 10 feet, but can be traced along its strike for at least 1,000 feet. The richest material contains upward of 16 per cent of graphite, but the average probably would be from 6 to 8 per cent, and the flakes are very minute -- from 0.04 to 0.15 millimeters in diameter. The deposits were prospected as early as 1881, but have never been worked.[23]

FOREIGN DEPOSITS

Graphite is found in practically all parts of the world. Seven countries - Austria, Ceylon, Chosen, Czechoslovakia, Germany, Italy, and Madagascar - furnish almost 85 per cent of the world output, but there are a dozen or more minor producing countries and innumerable localities where graphite, especially amorphous graphite, could be produced in quantity. Ceylon holds its place to a large extent because of the unique properties of its product, but elsewhere graphite mining is a matter of cost of production and transport to consuming market. The large output of Czechoslovakia, for example, is the result as well as the cause of a great local industry utilizing its products, and the same is true of Germany, Austria, and Italy. Districts such as Madagascar or Korea, remote from consuming centers, owe their exploitation solely to the fact that, due to favorable natural advantages and native labor, they can produce cheaply enough to overcome the handicap of having to ship their product halfway round the globe to market.

In the following notes no attempt is made to cover any but the more important present-day sources of world supply. Among the many localities which are not herein described, mention may be made at this point of Borrowdale (now exhausted) and other less famous places in England and Scotland; Berttula, Finland;[24] the Machakos District in Kenya (East Africa); the Victoria and Wankie Districts, Rhodesia; Uganda; Natal (near Ladysmith) and various other localities in British South Africa; Mt. Bopple, Queensland; Munglinup and Kendenup, Western Australia; Eyre's Peninsula, South Australia; Undercliffe, New South Wales; Senjen, Norway; the Departments of Suceara, Gorji, and Nehedinti, and the Carpathian Mountains, Rumania; Vittangi (amorphous) in Lapland and Norberg (flake) in central Sweden; Finland; Switzerland; the Belgian Congo and, just over the border therefrom, the Bambari District of French Equatorial Africa; Portuguese East Africa (Mozambique); Chile, Paraguay, and Uraguay; and Kirin, Shansi, and Honan (crystalline), and Chihli, Kiangsu, and Hunan (amorphous) in China.

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23 - Ball, S. H., Graphite in the Haystack Hills, Laramie County, Wyo.: U. S. Geol. Survey Bull. 315, 1907, pp. 426-28.

24 - Places noted in Finland, South Africa, Queensland, Australia, Rumania, Sweden, and China are described or mentioned by Imperial Mineral Resources Bureau. Graphite (1913-1919), London, 1923.

NORTH AMERICA

Canada - Graphite is widely distributed through Canada, but the only deposits that have been worked extensively are in southern Quebec and Ontario. Nearly all the Canadian graphite is of the small-flake variety, although up to 1908 a considerable amount of amorphous graphite was produced near St. John, New Brunswick, and a few tons of crystalline vein graphite (cf. Ceylon plumbago) was mined in 1917-1918 from Baffin Island. The typical deposits are either in sillimanite gneiss or in crystalline limestone. A less common occurrence is of the contact metamorphic type. In many years over one-half the total Canadian output has been furnished by the Black Donald mine at Calabogie in Renfrew County. This is the largest graphite mine in the Western Hemisphere and perhaps in the world, and it has been worked almost continuously since 1896. After being destroyed by fire early in 1928, the mill and various mine buildings were rebuilt. At the time of the fire 200 men were employed. The deposit, which is contained in limestone, averages 65 per cent graphite, ranging from 40 to 80 per cent. The proportion of coarse flake to dust is about one to three, the percentage being larger, however, near the footwall. Formerly the product was all sold for foundry facings, practically as mined, but since 1909 it has been concentrated by flotation. Six grades are produced -- three for lubricating purposes and three for foundry facings and stove polish.

Although the Ontario deposits are in limestone, those in Quebec are all in gneiss, resembling the Adirondack deposits in New York; most of the Canadian deposits, however, contain larger percentages of graphite.

Artificial graphite has been produced in Canada by the Acheson Graphite Co., a subsidiary whose works are situated on both sides of the national boundary at Niagara Falls.

Most of the Canadian graphite is exported to the United States, but there is a small local consumption for crucibles, foundry facings, and pencils, all of which are manufactured in Canada.

Mexico - The largest graphite producer in North America is the United States Graphite Co. which produces amorphous graphite from extensive deposits situated a little south of Hermosillo, Sonora, about 350 miles from the American border at the Santa Maria mine. The main graphite bed averaged 9 to 10 feet in thickness, but being highly folded it pinched and swelled so that in places it reached a thickness of 24 feet. There are at least six beds of which two or three are of good quality and of mineable width. The beds, originally coal seams, occur in sandstone near a contact with the granite which sometimes appears on the walls, and in small dikes which cut the beds. The graphite is exceptionally pure, and selected samples contain as much as 95 per cent graphitic carbon. A typical analysis shows 86.75 graphitic carbon, 7.60 per cent silica, .65 per cent iron oxide, and 5.00 per cent alumina.[25] During the last few years new deposits have been mined which are even richer than the Santa

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25 - Hess, F. L., "Graphite Mining near Colorado, Sonora, Mex.": Eng. Mag., vol. 38, 1909, pp. 36-48.

Maria and are much more accessible to the railroad. The output is all shipped to the mill of the parent company in Saginaw, Mich., where it is reduced to impalpable powder and employed for pencils, powder glazing, electrotyping, foundry facings, lubricants, stove polish, graphite paint, etc. This graphite is said to be superior even to the best Austrian graphite for the manufacture of pencils.

Deposits have been opened up in the same general vicinity by other American interests but have furnished very little production. Graphite was first discovered in Sonora in 1863.

SOUTH AMERICA

Graphite has been mined in a small way in Chile, Paraguay, and Uraguay, and deposits on the island of Chiloé and in the province of Atacaoma in Chile are said to be of potential importance. So far, however, graphite mining on a commercial scale in South America has been practically confined to Brazil, which has exported small quantities to the United States.

Graphite schists containing 10 to 11 per cent carbon have been employed locally for pencil-making. In Minas Geraes, notably in the Lafayette district, abundant supplies both of flake and of vein graphite have been noted; undeveloped deposits of flake graphite have also been located in the State of Sao Paulo, and in a number of places in Bahia amorphous graphite has been observed.[26] Veins 19 to 40 inches thick containing masses of graphite weighing hundreds of pounds are reported to occur at Emparedado, Bahia, but lack of transportation has prevented development.[27] A deposit about 10 miles from Aracoiaba, a station on the Ceará-Baturité Railway about 50 miles from the port of Ceará may be opened up in a small way.[28]

EUROPE

Austria - The former Austro-Hungarian empire produced annually over one-third of the world's output of graphite. Although the tonnage was substantially greater than that of any other country, the value was low, as the material was almost exclusively amorphous and most of it of low grade. About two-thirds of the pre-war output was derived from deposits situated within the borders of the new State of Czechoslovakia, but there are important deposits in the Alps of Styria and in the hills of Lower Austria which remain within the present republic. In the Styrian Alps the graphite occurs in highly folded slates and limestone. The most important mines are at Kaisersberg and Trieben, where there are from three to seven seams ranging from a few feet up to 30 feet in thickness and extending more or less continuously for 30 miles. The Styrian graphite, since it is derived originally from coal seams, varies greatly in

26 - Ferguson, H.G., Mineral Resources of the United States, 1918: Pt. 2, 1918, p. 244.
27 - U.S. Department of Commerce Reports, 40a, Sept. 25, 1917, p. 16.
28 - U.S. Department of Commerce Reports, Nov. 26, 1928, p. 553.

quality, some mines yielding soft earthy material and others a hard material grading into anthracite.

Graphite is produced in smaller amount in Lower Austria, where it is found in an area along the left bank of the Danube from 39 to 55 miles west of Vienna, probably an extension of the German district of Passau. Both crystalline and amorphous graphite have been mined. The former, however, is in very small flakes ("flinz"), rather sparsely disseminated in gneiss. Because the yield in flake is only 5 per cent and the flake is so small, this material is not mined under normal conditions, although during the war it was used for the manufacture of crucibles. Amorphous graphite occurs in graphitic slates which are found between the gneiss and mica schist. In the Muehldorf district a steeply dipping bed of impure graphite 35 to 130 feet thick occurs in crystalline limestone. As early as 1855 it had been mined to a depth of 500 feet in the Dunkelsteiner Wald. The graphite produced in Lower Austria, even after refining, contains only 60 per cent of carbon, but the rock is cheaply mined, and the yield is moderately high (30 to 50 per cent). The product is used principally for foundry facings but is also employed for electrical purposes and as a lubricant, more especially on the curves of street car tracks. In 1926 Lower Austria produced 4,500 tons of refined graphite; it could easily produce 7,000 tons. Of the two principal mines, the one situated at Muehldorf sells its own ore from an office in Vienna. The other at Horn sells through Heinrich Linn.

In Styria the crude ore runs from 40 to 95 per cent carbon, and needs only to be hand-sorted; some of it is shipped in lump form, but a considerable part of it is fine-ground and air-floated, and is then shipped in bags or wooden barrels. At the largest mine four grades are produced - one grade containing 45 to 65 per cent carbon; the next grade, 55 to 70 per cent; the next, 65 to 84 per cent; and the best grade, 82 to 84 per cent. The better grades are employed locally for making crucibles which although not so durable as those produced from crystalline graphite, are much cheaper. In 1926 the Styrian mines furnished 9,000 tons of crude graphite, but if the market permitted, they might yield four or five times this quantity. The principal mines are, in order of their recent production, the one of A. Miller's heirs at Treiben, Franz Mayr Melnhof's mine at Kaisersberg, and the Brockhues-Triebener graphite mine at Trieben - all owned apparently by one syndicate, Prager Handels Co., of Prague.[29]

Czechoslovakia - There are two graphite fields in Czechoslovakia, one in Bohemia and the other in Moravia. The former produces a little crystalline as well as amorphous graphite, while the latter produces only amorphous graphite. Although worked almost continuously since 1790 the mines of the Schwartzback-Krumau district in southern Bohemia, yield the bulk of the large output credited to that province, and there is said to be a large area of ground still undeveloped. The refined product resembles the graphite from Lower Austria in that it contains only 60 per cent carbon and is used chiefly for foundry facings and the cheaper kinds of pencils. The deposits are irregular lenses of graphite schist

29 - Petraschek, W., "Austrian graphite and its Sources": Eng. and Min. Jour., Oct. 8, 1927, p. 568.

interbedded with other metamorphic rocks. The Kollowitz district yields a slightly better grade of graphite but in much less quantity, and the Smojanow district yields a poorer grade, containing much pyrite. The Moravian deposits lie mostly near the Bohemian boundary; the most important are those of Altstadt, Kunstadt, Mueglitz, and Oels, all in the Brunn (Brno) mining district.

> "In the Alstadt district beds of impure amorphous graphite underlie an area of about 4 square miles. In the Kunstadt district, beds of graphite-schist are mined. In the Müglitz district, a bed of graphite-schist containing 32 to 42 per cent carbon and ranging in thickness from $1\frac{1}{2}$ to 65 feet, has been developed to a depth of over 300 feet. The graphite is used for manufacture of pencils and foundry facings. In the Oels district the graphite occurs in the same manner as at Müglitz, but the possible reserves are greater, and there are three beds 15, 18, and 35 feet thick, respectively. The graphite contains an appreciable amount of pyrite and is suitable only for pencils or paint."[30]

Germany - Although an important source of graphite, Germany has no surplus above its own large requirements. The imports in 1913 amounted to about three times the domestic output. This output is all derived from the Passau district in eastern Bavaria near the Austrian frontier. The economic value of these deposits is enhanced by the presence nearby of the famous Klingenberg clay, long considered the most suitable clay for crucibles. Crucibles were manufactured in Passau from local clay and graphite for the use of the alchemists in the Middle Ages. Before the war, Ceylon plumbago was used in large part even in Bavaria for making crucibles, and the local flake was employed principally for lubricants and foundry facings. In war time, however, Bavarian graphite was again used for crucibles, and the Passau district enlarged its output from 13,300 short tons of graphite in 1913 to 71,700 short tons in 1918.

The Bavarian graphite is of the disseminated flake type and occurs in lenses and pockets in gneiss and schist. Weathering extends to considerable depths; hence the ore is largely soft and sandy and is easily mined. Since the graphite content is fairly high, ranging from 20 to 30 per cent, production costs ought to be low.

France - Amorphous graphite has been mined in France for many years, chiefly in the Department of Hautes Alpes, close to the Italian border. The deposit is geologically similar to the Italian deposits. There are said to be undeveloped deposits also in the departments of Ariège and Rhone. Domestic supplies have been in the past, and will doubtless be in the future, a minor factor in the growing graphite-consuming industries of France.

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30 - Imperial Mineral Resources Bureau, Graphite, 1913-1919: London, 1923, p. 33.

Italy - Amorphous graphite, grading in places into anthracite, occurs in beds up to 10 feet thick in the Provinces of Turin, Genoa and Cuneo in northern Italy. The best refined graphite averages 70 per cent carbon, and is suitable for foundry facings and lubricants. There is a considerable surplus above domestic needs, which are rather small. This surplus formerly went largely to Germany, but more recently it has been sent to France and England. A little was sent to the United States. Before the war the Italian graphite used to be very cheap and the ocean freight was only $2.50 per ton.

Spain - Spain began producing graphite in 1915 from deposits near Almonaster, in Huelva Province. The graphite is of the flake variety and occurs in enriched lenses containing 10 to 20 per cent graphite in a belt of graphitic schists which extends from Alojao to Aroche and possibly beyond the Portuguese frontier. They have been developed more or less for a distance of 10 miles along the strike and over a width in places of half a mile. From 1750 to 1800, a considerable output of crucible graphite was made in Malaga Province, but the deposit was apparently exhausted as this province has subsequently produced only intermittently and in a very small way. For many years before 1913 Spain's meagre requirements of crucibles and other graphite products were imported.

Russia - The graphite resources of Russia are thus described in Mineral Resources of the United States for 1918:

> "Russia, particularly Asiatic Russia, possesses large reserves of graphite, but, so far as data are available, the production has been insignificant. The famous Alibert mine, near Irkutsk in eastern Siberia, formerly produced a considerable quantity of pencil graphite, and this graphite has also been used to a small extent in the manufacture of crucibles. The other important deposits, those of Yenisei River in eastern Siberia and those in the northern part of the Caucasus Province, probably consist of amorphous graphite. Some work was done on the Yenisei deposits in 1908 and 1909. In the extreme eastern cape of Siberia there are graphite deposits similar to those of the Nome Peninsula, Alaska. The availability of the Russian deposits depends on the possible future development of Russia as a manufacturing country, for, unless the deposits should prove of exceptionally high grade, difficulties of transportation would reduce their availability for foreign use.
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> "At the outbreak of the war the Morgan Crucible Co. established a crucible factory in Russia, and a large quantity of Ceylon graphite was imported. It is not known whether the clay was also imported or whether suitable local supplies were obtained.
>
> "The only deposit in European Russia concerning which information is available is that of the Dzimarsk district, in the Caucasus. Here graphite, presumably amorphous,

occurs in large beds or lenses with an average thickness of 1 meter and a known extension of 1,500 meters. The carbon content ranged from 30 to 60 per cent.

"Other occurrences are as follows:

"Pudosch district, near Onegasec, Olenetz government; Jelantschik district, southern Ural Mountains, granite boulders containing high-grade crystalline graphite; Zytomierc district, near Mecherzynce, Volhynia, near Mariupol, Jekaterinslav district, Ukraine, with a reported production of 2,000 tons (in 1913); Kumi Maggi, on the north shore of Ladoga Lake, Finland; Ekaterinburg; Miask; and Slatust. Graphite for foundry facings was recently mined at Boevsky, in the Ural Mountains."

Concessions have recently been offered by the Union of Socialist Soviet Republics for the exploitation of two graphite deposits, one in the Crimea and the other in the vicinity of Irkutsh in Western Siberia.

ASIA[31]

Ceylon - As measured by value of product, Ceylon has long been by far the principal graphite-producing locality. Competition from Madagascar combined with the curtailed demand for graphite crucibles for which Ceylon plumbago is particularly favored, has made heavy inroads into Ceylon's trade, but the island will continue for many years to be a large if not the principal source of world supply of high-grade graphite. The deposits were known as early as 1681; they were exploited before 1830 and have produced continuously since 1834. The increasing depth at which mining had been advanced has tended to increase costs, particularly in a country of such heavy rainfall, but reserves are still abundant and the graphite content of the rock is fairly high. The commercially important deposits, which lie in the Western and Southern Provinces, are all veinlike in character and were deposited along fracture planes in the rocks. Sometimes, however, the veins are so short as to be more correctly described as lenses. The width varies from less than one-eighth inch to several feet, and veins two-thirds of an inch wide or wider are said to be workable. In small veins the graphite usually forms an aggregate of platy needles set at right angles to the vein wall. In larger veins the graphite is ordinarily coarser though the needlelike structure may be developed for a short distance away from the walls. It is this needlelike material which when separated is classed as "needle lump," the highest priced of all Ceylon graphite because of its much desired fibrous structure. In addition to graphite, various gangue minerals may be present in the veins. Quartz and pyrite are nearly always present, and occasionally biotite, orthoclase-feldspar, pyroxene (usually augite), apatite, allanite, and rutile occur.

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31 - Siberian deposits are noted under European Russia.

Graphite is mined either from open pits or through vertical shafts connected with underground workings. A few mines are as deep as 500 feet, but most of them are less than 100 feet deep. With few exceptions they are owned and operated by the native Sinhalese. As a rule the mining methods are still crude and modern equipment is conspicuously absent. The material as it comes from the pits may contain as much as 50 per cent of impurities, mostly in the form of quartz and wall rock. It is carried in bags to a dressing shed where the impurities are reduced to 5 or 10 per cent by hand picking, after which it is packed in barrels and shipped to Colombo or Galle. At these ports it is unpacked and given the final preparation for market. The methods of "curing" employed by the different merchants vary somewhat. Usually the large lumps are broken with small hatchets by women. This releases the coarser impurities and the smaller lumps of graphite can then be rubbed up by hand on a piece of wet burlap and polished. The lump and chip grades from different mines ordinarily differ somewhat in size or texture and must be blended to meet the demands of the trade, a process requiring skill and long experience. The refining of poorer material is done less carefully. After being beaten up by wooden implements it is sorted into different grades. Occasionally it is washed in a tub or pit, the material placed in saucer-shaped baskets, and the graphite thrown out in much the same way as fine sand is removed in panning gold. Very fine material is dried and thrown up into the air so that the graphite is winnowed out and falls on the floor.[32]

The United States, which has frequently consumed more than one-half of the total exports, has been the largest purchaser of Ceylon graphite; the balance has been taken principally by Germany and Great Britain. Before the war 30,000 tons or more were exported annually, and the business is said at times to have afforded employment to as many as 50,000 persons. Since the war, however, the Ceylon industry has been much depressed; exports have amounted to less than half and often scarcely more than one-third the pre-war shipments. Various plans have been proposed to get back at least some portion of the markets that have been lost to Madagascar, but the consensus is that the chief requisite is lower costs. One proposal is to form a Plumbago Exchange through which the small mines can get into direct touch with the exporter and thus eliminate the profits of two or three middlemen who now intervene in most transactions. There are 17 exporters of whom about half are Ceylonese and half are European firms.[33]

Chosen (Korea) - The graphite industry of Chosen dates only from about 1903 when the first exports were made, but by 1913 production had advanced to more than 16,000 short tons and consisted almost wholly of amorphous graphite. Deposits of crystalline graphite have been developed, and in fact a few shipments of Korean flake have been made to the United States, but production of this variety has been sporadic and the product of poor quality. The amorphous graphite occurs in beds and irregular lenses which are found mainly in the

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32 - The deposits and methods of production in Ceylon are described in detail by Bastin in the chapter on Graphite in Mineral Resources of the United States, 1913, pt. 2, pp. 234-238.

33 - Mining Journal (London), "Ceylon graphite trade": Dec. 8, 1928, p. 1020.

central and eastern parts of the country. In one district (Kyeng-Sang) there are three seams having maximum mineable widths of 78, 18, and 48 feet, respectively. The crystalline graphite occurs in the northern part of the country in thin flakes, resembling the Alabama deposits in the United States. Korean graphite is used largely in Japan which has a very small domestic production. It is also exported to other countries, including the United States. A considerable part of the export trade is transshipped at Japanese ports. American capital has been employed in the exploitation of these deposits.[34]

French Indo-China - The existence of graphite deposits in various localities of Indo-China has been known for years, but it was not until 1917 that commercial exploitation began. The earlier work was down in Annam but veins were later opened up in Upper Tonkin on the banks of the Tam-Ty River near the Yunnan frontier. The ore as mined contains about 20 per cent carbon and the flake produced is said to be low in iron, almost free from sulphur, and otherwise suitable for crucibles, pencils, lubricants and other articles. The graphite occurs at the surface and is easily mined. A concentrating plant has been installed at the mine, and the output, all from one company, may be increased shortly to 4,000 tons annually. A little ore is being shipped to England and Germany and a few sales have been made direct to the United States but the bulk of the material is exported to France whence it may be reexported either crude or after treatment in a refining plant. Quotations c.i.f. French ports are approximately 2,300 francs (about $90) per metric ton for the 86 per cent carbon product.[35]

Exports were 8,000 metric tons in 1917, and were increased to 15,000 metric tons in the following year. This material came from Annam and much of it contained only 44 per cent graphite, although it doubtless could have been brought up to 80 per cent by more careful mining and sorting. In 1920 the industry collapsed and no exports were made from Indo-China for several years until in 1924 2,264 metric tons were exported. Later mining activity declined once more until the new deposits in Tonkin were opened up.

India - The State of Travancore in southern India formerly produced graphite similar to the Ceylon material; however, production became unprofitable with increasing depth, and there has been little or no output since 1913. A small amount has come from the State of Rajputana in the central part of the peninsula.

Japan - Japan produces a certain amount of graphite but not enough to supply the local market which depends partly on imports from Korea, Ceylon, etc. From 1900 to 1909 the output comprising both crystalline and amorphous grades fluctuated from 195,000 to 480,000 pounds. The occurrences are mostly in bedlike deposits, although there are a few veins. Of the bedlike deposits

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34 - U. S. Bureau of Mines, Mineral Resources, 1919: Pt. 2, pp. 199-200.
35 - Data furnished by E. A. Masuret, Office of U. S. Trade Commissioner at Paris. (Trade Notes No. 227.)

two varieties are noted:(1) flake deposits in Archean gneiss and (2) masses of amorphous graphite in Palaeozoic slates and Mesozoic shales. The flake deposits, which furnish the bulk of the production, are situated in the Provinces of Hida and Etchu in central Nippon. The deposits of amorphous graphite are found in the Provinces of Rikuzen in northern Nippon, and in Nagato and Satsuma in the southern part of the island. Graphite from Kataya in the Province of Kaga in central Nippon, said to occur as a vein deposit, is of high quality; it is used for pencil manufacture.[36]

AFRICA

Madagascar - Although the presence of graphite in Madagascar had been noted as early as 1838 the graphite industry of this French colony dates only from 1907 when 8 metric tons were produced and there was no appreciable exportation until 1911. By 1913, however, the annual output had grown to 8,000 metric tons and in 1917 it reached 35,000 tons. The graphite is of the flake variety and is similar in appearance to the best American flake although somewhat thicker. The occurrence is widespread throughout the graphitic schists and gneiss which comprise over two-thirds of the surface of the island. The ore lies in seams like coal, striking north and south with a general dip to the west. The beds are many miles in length and of considerable thickness - usually 30 to 65 feet and sometimes more. At operating mines the overburden never exceeds 6 to 10 feet. Mica appears principally in the upper portions and thus may be discarded in mining. The workable beds consist mainly of graphite and quartz and are so soft and decomposed that they can be dug out with wooden spades. The graphite content probably averages from 10 to 12 per cent. In the plateau region material containing 7 per cent of graphite is classed as ore, but in the coastal plain where transportation is better and there is more water for washing, the workable limit is 5 per cent.[37] On the other hand at one mine the graphite content is 40 per cent, and in some cases it is reported to be 50 or 80 per cent. In addition to flake graphite there is said to be more or less lump or fibrous graphite resembling that produced in Ceylon, but it has not yet appeared in any substantial quantity in the exports.

The deposits are usually worked by open cuts, although in a few cases they are opened up by short tunnels. Various crude processes are employed for raising the grade of the mine product. Sometimes the ore will be dumped into a mud box 3 feet square by 2 feet deep where it is spaded over in a stream of water. The overflow from this box -- suspended sand, mud, and flake graphite -- is passed over a 60-mesh screen. The flake remains on the screen and is gradually worked forward into a small box. In another cleaning process the mine dirt is beaten with a mallet or club, thrown into a square tub, and then agitated constantly; the graphite remains in the tub with the sand from which, after drying in the sun, it is separated by screening. In 1927 a concentrating plant

36 - Mining in Japan, past and present, Bureau of Mines. Japan, 1909, pp. 133-5 (digested by Bastin, E. C., in Mineral Resources of the United States, pt. 2, 1909, p. 27).
37 - Lavila, L., Graphite: First Commercial Fair (pamphlet), Tananarive, 1923.

was placed in operation by Lasnier & Co. at Tananarive.[38] Machinery for this plant was supplied by the Humboldt Co. of Cologne, Germany. The crude ore is ground in a ball mill, apparently in closed circuit with a classifier, and then treated in a 12-cell mineral separation flotation machine, the concentrates being filtered. The tailing was said to run 1 or $1\frac{1}{2}$ per cent carbon, and from 8/10 to 1 ton per hour of 87 to 88 per cent concentrate is obtained from crude material containing 40 per cent carbon. With richer ore a 90 per cent product can be made.

The leading factor in the Madagascar industry is the Maskar Co. which has a large warehouse at Tamatave and buys its ore from the many native operators. It is a subsidiary of the Morgan Crucible Co. of London, England, although in compliance with French law it is registered as a French company and has a French manager.

Formerly Madagascar graphite varied in quality from very good to rather poor, but under the auspices of the French Government a standard of 85 per cent was established, and material corresponding to this standard may be certified by an official stamp affixed to a lead seal. As early as 1922 a graphite commission was set up by the Governor General for the purpose of furthering the use of Madagascar graphite, and extensive advertising was done in the United States and Germany.

Madagascar has enormous reserves of readily available graphite of good quality. Costs of production some years ago were estimated at 450 francs, but in some instances they were as low as 250 francs per ton. In 1925 T. G. Hunter, a representative of the Ceylon Government, estimated the average as between 600 and 1,000 francs, and subsequently somewhat higher averages have probably been maintained. The labor shortage which has been a factor limiting production in peak periods doubtless could be remedied by introducing modern methods of mining and refining and even a small amount of modern equipment. Certain of the larger French consolidations have recently made some progress in this direction. In 1917 the graphite industry gave employment to several thousand white men and about 200,000 natives, but subsequently mining has had to compete with the growing of vanilla, maize, coffee, and other remunerative agricultural products. As in many native communities, larger wages, instead of attracting more labor, result simply in a reduction of the number of days worked. The average output of graphite per man per day is about 1 ton and the average wages are about 3 francs per day. Royalties vary; in one case reported by Mr. Hunter the royalty was about 100 francs per ton.[39]

Aside from labor conditions transportation is the principal handicap of the Madagascar industry. There are few roads on the island, and even when a new road is constructed it can scarcely be maintained except at prohibitive cost; often the product has to be carried to the warehouses on the backs of natives.

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38 - Madagascar refinery at Tananarive: Bull. Econ. Mens. No. 12 (Tananarive), Nov., 1927, pp. 44-46.

39 - Report of Plumbago Industry Committee, Colombo, XXXI: 1925 (Government Record Office), 15 pp.

Many of the mines, however, are within fairly easy reach of railroad or canal. Two men in a canoe, costing about 400 francs, can carry 3 tons so that even though the round trip lasts two or three days the carriage to railroad is often fairly cheap. From Tananarive to Tamatave the distance is 225 miles and the freight rate varies from 42.50 francs (about $1.65) per metric ton in carload lots (20 tons) to 50 francs per ton for shipments of less than 10 tons. Even after it reaches the warehouses at Tamatave, however, Madagascar graphite must be shipped a great distance before it reaches the markets of Europe or the United States. Except for an occasional shipment from Vatomandry -- somewhat further south -- Madagascar graphite is all shipped from the Port of Tamatave. According to Mr. Hunter carriage to French ports, both Havre and Marseilles, is by way of Messageries Maritimes and Havre Lines; the freight in 1925 was 165 francs or about $8 per metric ton. The ocean freight to London was 45 shillings ($11) and to New York 65 shillings ($15.85) per gross ton -- all in English bottoms. In 1925 the rate quoted to New York was $17.90 per gross ton.

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

GRAPHITE

PART III - UTILIZATION OF GRAPHITE

BY

PAUL M. TYLER

Circular No. 6123
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DEPARTMENT OF COMMERCE -- BUREAU OF MINES

GRAPHITE

PART III. UTILIZATION OF GRAPHITE[1]

By Paul M. Tyler[2]

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INTRODUCTION

The outstanding trend in the graphite market is the fast-growing demand for cheaper qualities of graphite and the declining consumption of the more expensive varieties. The manufacture of graphite crucibles for melting metals was one of the earliest applications of the mineral and for many years constituted by far the leading source of demand. In later years, however, and more especially since the Great War, the increasing demand for high-grade steel ingots and castings instead of enlarging crucible sales has permanently curtailed the domestic demand for crucibles because it has hastened the introduction of electric, oil or gas fired furnaces to replace the older types requiring pots to hold the metal during melting. While there has been a substantial increase in the use of graphite for miscellaneous refractories the total consumption for such purposes is too small to offset the shrinkage in demand for crucible making, and though it consumes the better and hence more expensive grades of crystalline flake graphite, it does not call so insistently for the still costlier Ceylon grades.

1 - The Bureau of Mines will welcome reprinting of this article but requests that the following footnote acknowledgment be made: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

2 - Assistant to chief, economics branch, U. S. Bureau of Mines.

CONSUMPTION OF GRAPHITE BY VARIOUS INDUSTRIES

The relative importance of the various outlets for both natural and artificial graphite is indicated by the results of the canvass made by the U. S. Tariff Commission of actual sales to ultimate consumers in the years 1923 and 1924. These figures fail to reveal the large quantities of graphite consumed in dry batteries, and probably there have been subsequent minor changes in the standing of the various consuming industries; but nevertheless these data are by far the most complete that have ever been made available. In fact, it is the first attempt to make a census of all consumers. Previous estimates have all represented merely guesses made by individual persons, and since much secrecy surrounds the operations of some members of the industry, such guesses may be far from accurate. The data obtained by the Tariff Commission, on the other hand, covered 27 of the 30 known manufacturers of graphite products. The three from which no information was obtained were of quite minor significance, and their omission is believed to have no material effect on the results of the canvass.

Before presenting the figures for the two-year period it is of interest to compare the standing of the leading consuming industries in each of the two years separately.

Percentage of total graphite consumed by the leading graphite products, 1923 and 1924

	1923, Per cent	1924, Per cent
1. Facings	43.5	51.5
2. Pigments and paints	16.	18.
3. Crucibles (steel and brass) .	15.	13.
4. Pencils and crayons	9.	5.
5. Commutator brushes.	8.5	5.
6. Stove polish.	2.5	1.5
7. Lubricants 3/	1.5	1.
8. Miscellaneous	4.	5.
	100.0	100.0

The indicated consumption for 1924 was 32,305 short tons as compared with 39,581 tons in the previous year, showing a reduction of 18 per cent. As compared with other consuming industries, the industries manufacturing foundry facings and paints seem to be more stable. In any event these industries accounted for a larger percentage of the total consumption in the later year when general business was moderately depressed. In other words, the reduction in consumption in the manufacture of these articles was less than it was in other consuming industries.

The average for the two-year period probably furnishes a better idea of the normal rank of the different outlets.

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3 - Lubricants, bearings, bushings, and dashpots combined consumed 3.2 per cent of the total in 1923 and 2.6 per cent in 1924.

Quantity, value, and kind of graphite consumed in the manufacture of various articles

(Annual average for the two-year period 1923-24 as reported to the U. S. Tariff Commission.)

Use	Short tons	Total value	Cents per pound	Total tons, per cent	Form of graphite used
Foundry facings............	14,279	$ 809,372	2.8	39.7	Amorph. and dust.
Pigments, paints...........	5,065	284,265	2.8	14.1	Amorph. and cryst.
Brass crucibles............	3,122	346,709	5.6	8.7	Crystalline.
Pencils, crayons...........	2,190	112,365	2.6	6.1	Amorph. and artif.
Commutator brushes.........	2,090	106,263	2.5	5.8	Artif. and amorph.
Steel crucibles............	1,073	166,318	7.7	3.0	Crystalline.
Stove polish...............	583	32,181	2.8	1.6	Amorphous.
Lubricants.................	451	121,064	13.4	1.3	All.
Bearings, bushings.........	365	100,492	13.8	1.0	No data.
Retorts....................	318	26,380	4.2	.9	Crystalline.
Shot and powder polish.....	100	5,700	2.9	.3	Amorph. and dust.
Packings...................	97	6,365	3.3	.3	Amorph. and cryst.
Ladle stoppers.............	95	9,766	5.1	.3	Crystalline.
Roofing dust...............	87	5,038	2.9	.2	Amorph. and artif.
Dry batteries a/...........	87	5,038	2.9	.2	Amorphous.
Boiler compounds...........	85	8,926	5.2	.2	Crystalline.
Dashpots, plungers.........	68	4,586	3.4	.2	No data.
Electrotype dust...........	47	6,111	6.5	.1	Amorph. and artif.
Electrodes b/..............	38	2,122	2.8	.1	Amorphous.
Finished products..........	30,240	2,159,061	3.6	84.1	Various.
Semifinished(not specified)	5,703	384,377	3.4	15.9	No data.
Total..........	35,943	2,543,438	3.5	100.0	Various.

a/ Obviously incomplete because the chief consumption in this field is of artificial and, in addition, at least two thousand tons of Texas flake are annually consumed in the manufacture of dry batteries.
b/ No data are available on the large quantity of artificial graphite used for electrodes.

Foundry Facings

Foundry facings are employed to give the surface of molding sands a smooth skin, and to enable the castings to be removed freely on cooling. In addition to preventing the sand from sticking to the casting, the facings make it possible to obtain a better surface. As graphite is highly refractory and slippery, it meets these requirements admirably. Occasionally the cheapest grades of graphite are mixed with molding sand itself, but except perhaps in certain parts of Europe the amount so used is trifling. In lieu of graphite, other forms of carbon, such as charcoal, coke, coal and gas retort carbon, may be employed; talc, either alone or mixed with graphite, is another substitute,

as are also silica, carborundum and a few other materials. The facings containing graphite or other carbonaceous matter are commonly called "blackings" to distinguish them from "mineral facings" which contain such materials as talc, and silica. Except for the universal requirement of extreme fineness, the preparation of facings follows no generally accepted standard; and the result is that there are probably at least as many formulas as there are manufacturers. Although the foundries almost never manufacture their own facings, a considerable number of foundry supply firms have suitable grinding and mixing equipment. The seven largest facing manufacturers furnish less than one-half the total output, of which the remainder comes from a much larger number of smaller firms.

For green sand molds, foundry facings may be dusted on the surface and then either "slicked" with a tool or the excess may be blown off; sometimes they are applied with a brush. For dry sand work they are applied wet with a swab; molasses water or some similar sticky liquor or clay is commonly added. Nearly all facings require a bonding agent in order to hold the graphite in place against the mold because graphite alone runs before the metal. The proper proportioning of the binder, usually clay, is important.

In addition to being mixed with other constituents in foundry facings, the graphite itself varies largely in purity. According to some authorities a very pure graphite is desirable even when it is to be diluted later with cheaper materials, but others contend that the silica and mica present in lower-grade graphite may be of actual benefit, causing the graphite to cling and to spread better over the surface of the mold. A great deal of dust containing only 40 to 60 per cent carbon is sold by refining mills to the foundry-facing trade, which constitutes virtually the only market for such low-grade material. Amorphous graphite -- Korean, Mexican, or domestic -- if purchased in crude condition needs to be dried and crushed before undergoing the usual grinding in tube mills, mixing, and air-floating.

Conflicting views exist as to the future of the foundry-facing business. Some feel that the larger employment of sand-blast, welding, and die-casting equipment in foundries will make less necessary the fine finish on the castings that graphite facings are designed to produce, and that this fact will cut down the demand for graphite for this purpose. On the other hand, the sand used for cleaning castings is expensive and the number of castings is steadily increasing.

Pigments and Paints

Even in its thinnest flakes graphite is perfectly opaque; since also it remains unaffected by sulphurous gases, acids, alkalies, and various fumes, it is employed extensively in paints, more particularly in protective paints for structural ironwork. Bridges, railroad cars, smoke stacks, boiler fronts, tanks, metal roofs, and other exposed surfaces, are commonly covered with graphite paint. Graphite invariably must be mixed with other pigments such as iron oxide, lead, or zinc compounds, as otherwise it tends to coagulate in the vehicle and spreads

under the brush into an excessively thin coating.[4] Silica or some similar material is also generally necessary to give the requisite tooth--both to enable a thicker film to be applied and to permit repainting. Paints containing too large a percentage of graphite present a satin-like surface which often can not be repainted satisfactorily, for succeeding coats curl up and peel because they can not adhere to the smooth film of graphite.

Both crystalline and amorphous graphites are used extensively in paints. Artificial graphite has also been employed by some manufacturers. Flake graphite has a better covering power than amorphous graphite but there is no demand in the paint trade for the better grades because most of the impurities naturally present in the cheaper grades of graphite, notably silica, actually make it more desirable, for they not only give tooth but also are supposed to add permanence to the paint film. In Michigan and elsewhere graphitic rock is ground directly into pigment, and the product often contains as little as 30 or perhaps only 25 per cent of graphitic carbon. Even after being ground, graphite pigments rarely are worth more than $3\frac{1}{2}$ or 4 cents, and some kinds of amorphous graphite pulverized to pass through 350 mesh sell for $40 a ton (1929). Anthracite coal is sometimes used in these paints.

Crucibles

Graphite crucibles are used in the production of malleable castings, small iron castings, crucible steel, brass and other copper alloys, and zinc castings, and for the melting of gold and silver. The crucibles are divided into two principal groups, (1) steel crucibles and (2) brass crucibles. Other metals are ordinarily melted in one or the other of these two types. The general shape of crucibles is that of an egg cut off flat at both ends. Steel crucibles are barrel-shaped except for the fact that although the bilge comes a little more than halfway up they are often a trifle smaller at the top than at the bottom. In proportion to their height they are generally smaller in diameter than brass crucibles, which differ also in being larger at the top than at the bottom. The notation of size is purely empirical and has been changed several times. Brass crucibles are rated according to capacity, assuming 3 pounds per number; thus a No. 70 crucible, which is a common size, has a capacity of 210 pounds of molten brass.[5] They vary all the way from a No. 12 (36 pounds) to No. 400 (1,200 pounds), and there is a growing tendency to use larger crucibles. Steel crucibles are numbered differently. They are made chiefly in two sizes. A No. 50 pot holds about 95 pounds and a No. 60 about 110 pounds.

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4 - According to Toch ("The Chemistry and Technology of Paints" by M. Toch. 2nd ed., New York, 1916, p. 101.)pure graphite paint will cover from 1,000 to 1,600 square feet to the gallon. The coat looks satisfactory to the eye but is too thin to last long. Although a mixture of silica and graphite produces good results, even it has too much spreading power. On the other hand, a six-year test of a linseed-oil paint containing 75 per cent ferric oxide and 20 per cent silica mixed with 5 per cent of 85 per cent graphite "proved itself to be as good a paint as can be desired for ordinary purposes."

5 - Owing to differences in specific gravity of the various metals and alloys, a No. 70 crucible will melt only 200 pounds of bronze but will melt 250 pounds of silver or 350 pounds of gold.

A different and larger type of graphite crucible is the retort used in the distillation of zinc in silver refineries, in the recovery of secondary zinc from galvanizer's dross, etc. Such retorts are often 40 inches high. Another type of crucible is designed to hold the molten lead or salts used for hardening and tempering small tools, files, etc. Still another is employed for dip-brazing bicycle frames. Numerous special shapes such as those used for annealing boxes and similar services are also made by the crucible manufacturers from the same ingredients that crucibles are made of.

The requisite qualities of a good graphite crucible are refractoriness, mechanical strength, good heat conduction, and long life. Refractoriness and the ability to withstand many heats involve satisfactory resistance to chemical action as well as to high temperatures. The graphite in the wall of a crucible begins to oxidize at a temperature somewhat under 700° C.; its rate of oxidation from the outside varies with the composition of the furnace gases and on the inside with the composition of the material being melted.

Ordinarily approximately 50 per cent of the crucible mixture is graphite, the remainder being chiefly clay and sand or grog. Sometimes the percentage of graphite is much less. The nature of the other ingredients is obviously significant; the clay in particular seems to contribute as much as thegraphite to the qualities of the product. Care in manufacture and proper curing vitally affect the serviceability, and the nature of the service and proper handling likewise enter into the final result. So many factors are involved that it is unwise to be too dogmatic about just what does or what does not constitute the best practice in manufacture. The extensive literature that has grown up about this subject is full of conflicting statements out of which may be gleaned only the facts that good clay as well as good graphite must be employed and that proper care must be exercised throughout all the various stages of manufacture including drying, storage, transportation, preheating, handling in the furnace, actual melting service, and cooling between successive heats. If a suitable glaze can be formed on the surface of the crucible, the graphite will be protected from oxidation and thus last longer. As regards practical performance, the following statement by Searle[6] give some information on British practice:

> "It is clearly impossible to give any accurate figures showing how many times a crucible may be used. Some men can use a crucible eighty times or more for cast iron, whilst others can only use a similar crucible half a dozen times. Similarly, in the melting of alloys, steel, and all other purposes for which crucibles are used, the manner in which the tongs fit and the handling of the crucible being much more important than is usually imagined, the following figures represent fair averages:-
>
> For brass, a crucible should serve 70 to 100 times.
> For bronze, " about 50 times.
> For iron, " 70 to 90 times.
> For steel, " 6 to 10 times."

6 - Searle, A. B., Refractory Materials, Their Manufacture and Uses: London, 1917, pp. 304-25.

Other authorities estimate 60 melts for brass or copper and from 10 to 15 heats for steel.

Kinds of Graphite Used. Various kinds of graphite have been used for making crucibles. Passau (Bohemian) flake was used first, and for inferior crucibles a considerable admixture of amorphous graphite is now used in central Europe, simply because it is cheap. Ceylon plumbago has been universally recognized as the variety best suited for crucible use. American, Canadian, and Madagascar flake have all been used with good success, but there is a wide divergence of opinion as to whether as good crucibles can be made commercially from 100 per cent flake graphite as from the usual mixtures containing a large proportion of Ceylon plumbago. Flake graphite was substituted to a large extent for Ceylon grades during the war, but even France, where Madagascar flake exclusively was used in crucibles during the later years of the war, has slowly reverted to the use of at least a small percentage of Ceylon plumbago. Ceylon graphite has much greater bulk with respect to its surface area, as the individual particles are mostly wedge-shaped or rod-like, and hence it requires proportionately less clay as binder than, for example, the thin flakes of domestic graphite. It is also said to be more nearly free from undesirable impurities such as mica and pyrite. On the other hand, some authorities contend that a large number of thin overlapping flakes is preferable to a smaller number of more or less angular fragments. Although the latter type furnishes a better bond, the former provides better for the slight slip and readjustment within the wall of the crucible as it expands or contracts, and thereby reduces the liability of cracking. Moreover, these flakes, according to tests made by the Bureau of Mines, actually resist oxidation longer, as only the edges are exposed to the oxidizing influences.[7] Abundant evidence has been offered in support of both sides of this question, but the weight of opinion among domestic crucible manufacturers as shown by their purchasing policy is overwhelmingly against employing more than about 25 per cent flake. Most of them prefer to use Ceylong lump or chip for 80 per cent or more of the total graphite admixture. Formerly the remainder was mainly Ceylon dust, but now Madagascar or domestic flake is used instead.

A. V. Bleininger[8] places the comparison on the basis that a dollar's worth of Ceylon graphite yields more crucible value than a dollar's worth of flake, adding that if Ceylon graphite could not be obtained, the production of metal would not be diminished in any way, as "we could get along very well with flake and amorphous graphite, furnace carbon, and coke." He gives the following figures for the actual volume of the different kinds of graphite after thorough shaking:-

	Volume occupied by 100 g., c.c.
Ceylon graphite	90.7
Canadian graphite . . .	119.6
Alabama graphite. . . .	152.0

7 - Thiessen, R., "Structure of graphite in relation to crucible making:" Jour. Am. Ceramic Soc., July 1919, pp. 508-542.
8 - Bleininger, A. V., Can. Chem. Jour., October, 1918, p. 253.

The specifications for crucible graphite rarely call for much over 90 per cent graphitic carbon, but the nature of the remaining 10 per cent is of much importance. Of the ordinary impurities, mica, which is difficult to remove, is one of the most common and also one of the most injurious; it fuses readily, making pin holes in the walls of the crucible. Carbonates such as limestone are undesirable because they decompose when heated, leaving shrinkage cavities. Sulphur, usually in the form of pyrite, is present in small amount in most Canadian, American, and Bavarian flake and, to a less extent, in Ceylon graphite; only Madagascar flake is virtually free from it. Sulphur is decidedly deleterious, as it not only causes blowholes that may result in a porous condition but also may contaminate the metal that is to be melted.

The size of the graphite particles that go into the crucible mixture is commonly from 20 to 90 mesh; the lower limit of size varies in the specifications of different manufactures from 86-mesh (No. 8 silk bolting cloth) to 125-mesh (No. 12 cloth). Most crucible manufacturers, however, further refine their flake, ordinarily grinding it between buhrstones and then screening out the badly broken flake and dust. This dust, since it is worth but a fraction of the price of good crucible graphite, is kept at a minimum, and manufacturers are reluctant to buy graphite containing too great an excess of gritty impurities that would unduly destroy the particles of graphite. The dust could be disposed of profitably in the form of lubricants, paints, or other finished products, but the amount produced is too small to interest most crucible manufacturers in the production of a diversified line of graphite articles. Most of their fines can be used in stoppers and nozzles which are generally made in the same factories with crucibles.

Production Data. Separate statistics for the production of graphite crucibles are not available. The Federal census reports "crucibles of all kinds, including glasshouse pots, made largely from fire clay and plumbago (graphite)," but these figures cover "miscellaneous products, normally belonging to other industries, made as secondary products by establishments engaged primarily in the manufacture of crucibles." The value of the miscellaneous items was calculated to be only $75,257 in 1925, and, although there are no data for such items in previous years, it may be assumed that the trend of the industry is revealed by the following table:

Production of crucibles, 1919 to 1925
(Data from Federal Census)

	1925	1923	1921	1919
Number of establishments......	11	13	17	22
Wage earners (av. number).....	284	516	412	848
Horsepower...................	1,476	1,647	(a)	2,714
Wages........................	$ 380,783	$ 626,678	$ 495,973	$ 923,287
Cost of materials............	$ 754,817	$1,226,159	$ 827,288	$2,233,072
Value of products (b)........	$2,091,518	$3,467,816	$1,969,930	$5,293,688
Value added by manufacture....	$1,336,701	$2,241,657	$1,142,642	$3,060,616

(a) No data.
(b) Does not include crucibles made as secondary products in other establishments.

For 1925, the five establishments in Pennsylvania gave employment to 40 per cent of the wage earners, consumed 56 per cent of the materials, and produced over 57 per cent of the total value of the products. Of the remaining six establishments, two were in New Jersey and one each was in Connecticut, Indiana, Massachusetts, and New York.

The above figures do not cover the large output of crucibles made as secondary products by establishments classified in other industries. The values of such crucibles do not quite follow the same trend as those made as principal products: 1925, \$1,869,105; 1923, \$809,971; 1921, \$564,398; and 1919, \$1,570,579. The difference is too slight, however, to alter the general downward tendency.

Imports. Imports of graphite crucibles have been reported separately since the Tariff Act of 1922, but they have been quite small.

Imports for consumption of graphite crucibles, 1922-1928

Calendar year	Number	Value	Unit value
1923	1,655	\$ 459	\$0.28
1924	- -	- -	- -
1925	3,100	548	0.18
1926	9,513	11,780	1.24
1927	2,583	1,812	0.70
1928	4,170	1,441	0.34

Exports. Exports are of some importance to the industry, as American crucibles are used extensively in Canada, in Mexico, and to a lesser extent in other Latin-America countries. The extent of the export business is not known, however, because the figures for graphite crucibles are included with those of other crucibles. Although the unit values of the graphite crucibles are probably greater than those of other crucibles included in the export classification, the range in sizes and prices of both plumbago and clay crucibles is so great that price alone does not afford a statistical basis for separating the figures. In 1927 the total exports were 558,000 crucibles valued at \$108,857. Of these, 356,822, or 64 per cent of the total number, were exported to Mexico. The value of the crucibles exported to Mexico, however, was only \$31,286 or 29 per cent of the total value.

Other Graphite Refractories

The refractory properties and other characteristics of graphite that render it desirable for crucibles also its use in a variety of other accessory articles used in smelting and foundry work, such as crucible covers, pouring nozzles, crucible rests and stools, funnel or extension tops, skimmers, phosphorizers, ladle stoppers and stopper sleeves, pyrometer sleeves, furnace bricks and doors, annealing boxes, and case-hardening containers. Special mixtures are used in the so-called "graphite blocks" employed in flattening window glass. Graphite also enters into the composition of certain refractory cements as a lubricant, making them flow more freely under the trowel.

The output of several of these items - notably that of ladle stoppers, which are used at every iron and steel works and at most foundries and brass plants - is large and growing. For some of the articles Ceylon plumbago is used sparingly, if at all, and for a few of them the cheaper grades of domestic graphite such as No. 2 flake may be employed. In general, however, the manufacture of these miscellaneous refractories is undertaken as a branch of the crucible business, and are made by the same firms and from the same materials.

Pencils and Crayons

One of the earliest and by far the best known use of graphite is to make marks on paper. The name "graphite,"it may be noted, is derived from the Greek word meaning "to write." In the sixteenth century crayons were being made in England from the graphite found in the Borrowdale mine. For many years solid blocks of the natural material were simply cut into sticks and sold as such. Later they were cut somewhat smaller and protected by a wooden holder, but it was not until late in the 18th century that the present method of making pencil "leads" from an artificial mixture of graphite and clay was adopted. In its early days ownership of the Borrowdale deposit in Cumberland constituted a monopoly of such importance that after a time the government assumed control of the operation. Finally it prohibited the export of this graphite except in the form of pencils. The mine was worked only for a short period -- usually six weeks -- each year, but the product was auctioned weekly on the London black lead market, selling usually for 30 to 40 shillings a pound and choice qualities brought sometimes as much as 140 shillings. Despite all measures for conserving the supply, the richer parts of the mine became exhausted. Poorer portions of the deposit and the great heaps of waste were then picked over and eventually the lean rock was ground and washed, but the material so obtained was of much poorer quality. Various schemes were tried in the effort to bind together the small fragments so as to produce a cake that could be cut up in the same manner as the natural lumps. Most of the available binders proved very unsatisfactory, although some success followed the use of rosin mixed with a little wax or tallow.

In 1795, Conte in France and Hardmuth in Austria both began to make pencils very much in the modern way. The function of the clay is not solely that of a binder. It acts as a vehicle rendering the graphite plastic so that it can be molded readily, and it affords a means of regulating the hardness and of maintaining uniform quality, both of which were impossible with the natural block graphite. Except for the cheaper pencils, a mixture of graphites is generally considered better even than the best single variety. The product of the Borrowdale mine, which was closed in 1833, was exceptionally suitable material; the mineral occurred in pipes, stringers, and nests in association with a dike of altered diorite intruded into volcanic ash.[9] Some of it may still remain, but prospecting in 1878 seemed simply to confirm the fact that this famous deposit was economically exhausted, even tho such little of the product as was obtained was said to have sold then for as much as $10 a pound.

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9 - This statement is from Spence who refers to Mem. Geol. Surv. Great Britain, vol. V. 1916, p. 25, and to Haenig, A., Der Graphite, 1910, p. 41.

Some 15 years after the English mine ceased to operate, similar material was discovered in the Sayan mountains west of Irkutsk in Siberia. This became the well-known Alibert mine which supplied the Faber pencil factory at Nuremburg with all its graphite for many years. After a time, however, this mine likewise was exhausted or abandoned. Since then almost all pencil makers instead of depending upon only one mine have been using two or more entirely different types of graphite, and the results are so satisfactory that the fancy prices formerly paid for the product of these two famous mines could not be obtained again.

Crystalline graphite can not be used alone because it slips over the paper, making only a faint mark; moreover it is difficult to grind it fine enough. However, a particular grade of Ceylon graphite has been used to a considerable extent in blends because it does add smoothness. Bohemian grades have a reputation for blackness, and somewhat more recently the Mexican graphite produced by an American company has found its way into European pencil mixtures and has been used successfully in the United States without much if any admixture of other graphites. In addition to color, freedom from grit, uniformity of grain, and softness are important qualifications for pencil graphite. Artificial graphite, because it often contained small particles of carborundum, was for a long time unpopular with pencil makers, but a certain amount of it is now used for pencils.

Besides graphite and clay, other substances such as sulphide of antimony, lamp black, or finely divided metallic lead may be added. Sometimes the pencil leads are boiled in wax or tallow in order to toughen them or to remove grit. The proportions of the various ingredients vary considerably even for pencils of similar quality. The percentage of graphite is ordinarily higher for soft pencils as the clay tends to make the lead both harder and less lustrous. As a general rule the graphite amounts to 50 per cent or less of the mixture, but it may be considerably more; Cirkel,[10] for example, gives the following recipe as having been employed:

	Parts
Graphite........	30
Clay.............	9
Stibnite.........	9
Tallow..........	1
	49

and for very hard drawing pencils:

Graphite........	36
Clay............	18
Stibnite........	8
Lampblack.......	2
	64

10 - Cirkel, F., Graphite, Its Properties, Occurrence, Refining, and Uses: Canada Dept. of Mines, Mines Branch, 1907, p. 263.

Proper mixing is essential. The clay is carefully selected and washed very thoroughly to eliminate any possible grit, and the mixture is worked and reworked until it can be looped and coiled and even tied into knots. The dough then goes to a hydraulic press and is extruded through holes of the proper size. After being cut to length the leads are packed in a crucible with powdered carbon and burned at a definite temperature (usually 1,500° to 2,000°F.). If heated too rapidly the leads will warp or spring, rendering them useless; they can not even be ground up and reburnt.

The whole process of pencil manufacture is described by Spence as follows:[11]

> "The clay is ground dry, water-floated, and settled in tanks. Graphite and clay are then mixed in the required proportions and further ground between buhrstones in a closed tank. This grinding is wet, and according to the degree of fineness required, lasts from two weeks to three months. The sludge from the buhr mill is then passed to a filter press or vacuum filter, preferably the latter, since the cloth screens of the former tend to clog. The product from the filter is kneaded by hand to the required consistency, and the dough is fed to hydraulic presses which force it, under a pressure of 2,000 pounds per square inch, through dies of the diameter of the finished lead. The lead issues from the die in the form of a continuous string, and the die head, being mounted on a toggle joint, the string is coiled as it exudes from the die and is caught in a shallow metal dish. The dish is removed when full, and the lead is uncoiled by hand and pinched off into lengths, each equal to three pencil lengths. At this stage, the material is quite tough and pliable and can be readily handled without breaking or deformation. The lengths of lead are laid between boards and allowed to air dry, after which they are cut by hand into pencil lengths and arranged in bundles in graphite crucibles or boxes. They are then placed in a kiln and baked for several hours at a temperature of 1,500° to 2,000° F., after which they are ready to be placed in the wood casings. An intermediate drying may take place before the final baking, and is effected in iron boxes in a hot air chamber having a temperature of about 150°F.
>
> "The wood casings consist of cedar blocks, grooved to receive the leads, and measuring 7¼" x 2¼" x 3-3/16." Each block is provided with six grooves and after insertion of the leads, the blocks are dipped in glue and clamped together in bundles of a dozen or less, and allowed to dry. Finally, the glued blocks are cut up into individual pencils, which are then trimmed, sandpapered, painted, varnished, and stamped.

11 - Spence, H. S., op. cit.,pp. 146-7.

"To impart the necessary strength, the softer grades of pencils have leads of greater diameter than the harder grades, in which the larger proportion of clay used serves the same purpose."

The value of pencils manufactured in the United States first exceeded $1,000,000 in the year 1890. Although the domestic industry continued to grow quite rapidly, pencils, particularly drawing pencils of the better quality, were imported into the United States in large quantities before the war, principally from Germany and Austria. When supplies from foreign sources were largely cut off by the outbreak of hostilities, the American industry expanded rapidly. The Armistice found it in very favorable condition. While the imports have recovered to the point where their value amounts to $600,000 to $700,000 annually, or practically the same as pre-war figures, exports have grown to more than three times the imports and domestic business has increased to very large proportions. The American pencil industry having outstripped that of Czechoslovakia, has now become larger than that of any other country, with the possible exception of Germany.[12] A large part of the increased business of American pencil manufacturers, both in domestic and export markets, is the result of the extensive use of automatic magazine pencils.

With respect to raw materials the American pencil industry is well-situated. Previous to 1900 it used Bavarian or Bohemian graphite and German clay, but later it began to use Mexican graphite and, when the necessity arose, domestic clay. For a time after the war, it derived practically all its raw materials from the North American continent, but one or more manufacturers have since reverted to the use of Klingenberg clay. For the better pencils cedar wood is preferred and even foreign manufacturers draw their supplies of such wood chiefly from the southern United States.

The American pencil industry, which is centered mainly in Illinois, New Jersey, New York, and Pennsylvania, produces annually, according to recent figures of the Federal Census, products valued at approximately $25,000,000. A little more than one-half of this total represents wooden pencils and about $1,000,000 annually represents pencil leads for use in mechanical pencils.

Commutator Brushes

Carbon or graphite brushes have largely displaced metal brushes for electric generators and motors because, due to their being harder and having more electrical resistance, they provide a better contact and otherwise reduce sparking. With metal brushes it is difficult to keep the surface of the commutator smooth, as it does not wear down evenly. Graphite is used in most varieties of brushes, but it is usually mixed with other ingredients. Numerous grades are on the market which are composed wholly of graphite and little binding material, but they have very limited application. Coke, lamp black, or metal powders such as bronze, brass or copper are mixed with a graphite in varying proportions. Metal gauze brushes filled with graphite were used formerly, but they have been

12 - European manufacturers, however, have regained much of the business they lost during the war when Japan and the United States were the leading exporters.

largely superseded by metal graphite brushes in which the metal enters in the form of powder. These various mixtures after being made up in their proper proportions are molded under high pressure and baked to carbonize the binder. Sometimes they are electroplated and the larger ones may have a "pig tail" soldered or bolted to them.

In the United States amorphous and artificial graphite are employed, but certain foreign brushes are said to contain Ceylon or flake graphite ground to 100 mesh.

Graphite is added to electrical brushes for its lubricating and conducting properties. The ordinary carbon brush contains very little graphite and is composed mainly of amorphous carbon, usually petroleum coke. Ordinarily the brushes are impregnated with some additional lubricant in the form of grease in order to cut down abrasion, reduce friction, and create a little higher contact resistance, thereby improving the commutating properties. The carbon brush is medium hard and being moderately cheap is considered a good general-purpose brush. Brushes containing a somewhat larger proportion of graphite and less coke are known as graphite-carbon brushes. Such brushes are self-lubricating and moderately soft, but while they have a high current-carrying capacity, the contact drop and hence the commutating properties are not entirely satisfactory. This type of brush is used extensively on industrial motors, railway and mining motors, and on moderate speed generators; special grades are employed for light service, such as for magnetos and fan motors of both direct current and universal types. Graphite brushes are soft and have little mechanical strength, but being nonabrasive they are much favored for service requiring quiet operation. They may be used in high-speed work, automobiles, lighting generators, electric vehicle and battery locomotive motors, battery-charging and other low-voltage generators, in slip ring machines, and occasionally as lubricating brushes on electroplating generators.[13] Electrographitic or graphitized carbon brushes, made in electric furnaces at extremely high temperatures from petroleum coke or other forms of amorphous carbon, are suitable for a variety of uses. Metal graphite brushes are used extensively in automobile starters, plating generators, and railroad signal motors, as well as for slip ring machines of large power.

Stove Polish

Amorphous graphite, chiefly Korean and Mexican, is used in stove polishes. Carbon black may be added to improve the color, particularly if impure grades are used. Clay, rosin, soap, or asphalt may be used as a binder for the cake polishes; for the liquid polishes, gasoline, various oils, ammonia, or a plain water are used as vehicles. In all cases the graphite has to be ground fine.

13 - Kalb, W. C., "Application of brushes to electrical machinery:" Power, vol. 49, No. 7, Feb. 18, 1919, p. 241.

Lubricants

Graphite is an excellent lubricant and retains its lubricating properties at all temperatures and under conditions that cause oil or grease to decompose or lose body. It adheres readily to metal and not only fills up the pores but also forms a protective film which can not be squeezed out even at high bearing pressures. The field for solid lubricants has increased rapidly in late years due to the increased use of heavy machinery. While mica, talc, sulphur and other substances are used for similar purposes, graphite is by far the most widely used and in many cases the best solid lubricant. The total tonnage of graphite used in lubricants, however, is quite small.

According to Spence:-[14]

"For ordinary lubricating purposes in loose, open bearings, gears, slides, etc., graphite is commonly mixed with oil or grease, and there are a variety of such compounds on the market, many of them designed for work under special conditions, such as exposure to salt water, acids or alkalies, in dredges, pump plungers, winches, mining machinery, etc., or at different temperatures, where varying degrees of viscosity are required.

"In cylinder lubrication for steam and gas engines, compressors, etc., modern practice is to feed dry flake or powdered graphite in addition to grease or oil. For this purpose, numerous special lubricators have been devised. In marine engines, the use of graphite for cylinder lubrication is especially advantageous, as by its use, the amount of oil finding its way into the condensers and boilers is materially reduced.

"Graphite is also used in pipe joint compounds, for lubricating and sealing the threads and flanges of steam, water, gas, oil and air pipes, and for bolts, nuts, studs, caps, boiler plugs, manhole plates of boilers, gas retort doors, metal gaskets, etc. Such compounds replace, and are claimed to be superior to, red or white lead. As in the case of graphite paints, superior merit is claimed for each of the three types of graphite -- natural flake and artificial and natural amorphous -- for use in lubricating oils and greases.

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"Dry graphite powder is used for lubricating the actions and bridges of pianos and organs, and in short, any wooden or other surface when the use of oil or grease might be detrimental, such as in textile machines. It is employed, also, in typesetting machines, to give the space bands, channel plates, etc., a dry, smooth surface.

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14 - Spence, H. S., op. cit. pp. 155-6.

"To minimize water friction, yacht and launch bottoms are sometimes dusted with graphite after a preliminary light coat of varnish or shellac has been applied, and when dry, are polished with cloths or waste. The graphite used for this purpose is sometimes termed 'potlead.' "

Various so-called self-lubricating bearings and bushings are on the market. Some of them are hardwood impregnated with a lubricating compound containing graphite. Another type is a bronze casting having grooves or slots filled with a graphite mixture and then baked and afterwards broached to the proper size. The former are used extensively for light duty parts -- for shackle pins in automobiles and for loose pulleys and bearings on light textile machinery. The latter are especially adapted to the requirements of trolley wheel and windmill bushings and to other moderately heavy service in places difficult of access where lubrication by ordinary means might be neglected. Antifriction alloys into which graphite is incorporated during the melt have been produced but do not seems to have worked well. Practically pure graphite consolidated under a pressure of 20 tons per square inch is offered in various forms under the name of "morganite" by the Morgan Crucible Co. of London, England. Bearings machined from this material are claimed to be mechanically strong and possessed of the lowest possible coefficient of friction.

Oildag, the well known lubricating compound, of the Acheson Graphite Co. is a colloidal suspension in oil of deflocculated graphite said to be "graphite reduced to the molecular form, the finest state of subdivision" (by a chemical process using tannic acid).

The lubricating qualities of graphite result in its employment in various packings, particularly for steam engines and pumps. Plungers made of graphite or graphite-impregnated metal are used in pneumatic tube service equipment in stores, in dashpots used in electric train equipment, in the dashpots of arc lamps, and for other apparatus of this general nature where it is necessary to have plungers lubricated but where it is not desirable to use oil. Efforts are being made to market a very pure (99+%) natural graphite for lubricating automobile engines; the finely powdered material can be introduced directly into the cylinders through the carburetor.

Graphite Electrodes

The production of electrodes has been increasing rapidly with the rapid expansion in the use of electric furnace and electrolytic processes. Electrodes are of two principal types -- graphite and amorphous. There are two types of amorphous electrodes -- petroleum coke and anthracite. The domestic production of all classes was estimated by the U. S. Tariff Commission in 1921 as amounting to $10,000,000 or more annually. Although a tonnage business, the electrode manufacturing industry is one requiring careful control and constant inspection.

Coke electrodes are employed principally in the manufacture of aluminum. Anthracite electrodes are used principally in the production of electric steel, ferroalloys, calcium carbide, and similar electric furnace products.

The graphite electrodes, used chiefly in the electrolytic industries (notably the manufacture of chlorine and caustic soda) and in small and medium sized electric furnaces, are not made from natural graphite. They are more properly called graphitized electrodes. Attempts to produce satisfactory electrodes out of powdered graphite and a binder have generally failed. Tar and other binding agents have been employed, but electrodes made in this manner lack sufficient strength and soon break.

The three types of electrodes differ in their properties and, therefore, one type can not ordinarily be substituted for another. The properties which must be considered in choosing the proper electrode include the following: Specific gravity, mechanical strength, hardness, density, sonority, electrical and thermal conductivity, the purity and the nature of the ash produced.

Carbon electrodes are made from some form of carbon mixed with tar or pitch, molded and finally baked to drive off volatile matter. Retort carbon and petroleum coke have been largely used and are still employed wherever the purity of the electrode is important as in the manufacture of aluminum which, since it requires a larger consumption per unit of product than any other large use, requires an electrode that will yield little or no ash but that will contaminate the product. For most electrical smelting where mechanical strength, electrical conductivity, and cheapness are more essential than extreme purity, anthracite is now generally used. Graphitized electrodes are relatively pure, have good conductivity, and resist oxidation better than other types. They have the further advantage that they can be machined. In size they range from one-sixteenth inch to 12 inches or more in diameter.

Dry Batteries

Graphite is mixed with the manganese-dioxide in dry batteries to give conductivity. Granulated carbon was formerly used for this purpose, but as graphite is a much better conductor it has largely displaced amorphous forms of carbon for this purpose. Partly for this reason but more particularly because of the enormous increase in the number of dry cells produced, the dry battery industry, formerly an almost insignificant factor in consumption of graphite, has recently become one of the leading outlets. In 1928 it appears to have absorbed 50 to 75 per cent of its domestic output of crystalline flake, a much greater quantity of artificial graphite, and also a little amorphous graphite. For 10 years before the war, the Census figures showed an increase in domestic production from less than 5 million in 1904 to over 71 million in 1914. These figures appear insignificant, however, in comparison with enormous advances made since 1920. In 1927, the number of dry cells (including 6-inch radio B and C, and flashlight batteries) reported by the Census was 718,641,735 and the value of the products (including certain parts and supplies) was $48,986,860.

The use of amorphous seems to be on the decline, being displaced by more expensive varieties. For this purpose Texas flake seems to be much superior to Alabama flake, but it has to compete with artificial graphite which is said to be equally suitable. The largest producer of dry cells now controls the domestic production of artificial graphite. Very little foreign graphite is used in dry cells.

Miscellaneous Uses

Electrotyping graphite is very finely powdered, and is ordinarily made from either very pure amorphous dust or artificial graphite. This graphite is used in two ways. First it is dusted upon the forms which are polished so that the wax mold can be stripped clean, and then it is used once more for covering the mold, so as to form a conducting surface when immersed in the plating bath. In boiler compounds crystalline graphite acts as a lubricant, working its way into the scale and loosening it and preventing new scale from adhering to the metal. Either as a lubricant or simply as a filler it enters into steam and engine packings, hard rubber compositions, hemp rope, wire rope and cable, rubber valve discs and washers, etc. It is used as a dusting agent on certain kinds of prepared roofing to prevent sticking. The polishing of shot and the glazing of black and smokeless powder require a substantial amount of amorphous or dust graphite. One of the functions of the graphite used with explosives is to act as a lubricant, permitting the grains of explosive to flow freely without danger of spontaneous ignition. A somewhat similar use is for treating tea leaves and coffee beans. The graphite film is harmless and it protects the products from moisture and improves their color and general appearance.

GRAPHITE SUBSTITUTES

Because of its employment in crucibles used for melting many of the metals needed for the manufacture of munitions, graphite has been considered an essential war mineral and it fills an important place in the peacetime life of the nation. For almost all of its applications, except in crucible manufacture, substitutes can be found for graphite, many of which are quite as good and almost as cheap. Even the use of crystalline graphite in crucibles can be avoided by melting in electric furnaces, and, as has been noted elsewhere, the trend has been rapidly in that direction. Crucibles themselves can be made with various proportions of graphite or wholly without it; also, instead of the widely preferred Ceylon lump, flake graphite can be substituted with little if any detriment, and in Central Europe fairly good crucibles have been made for centuries with amorphous graphite. In general, then, the use of a particular kind of graphite for a particular purpose is mainly a matter of convenience and price. From a practical standpoint this statement may appear to be too strong, but the facts are nevertheless essentially true. This becomes apparent if we analyze the principal uses one by one.

Graphite dustings can be at once dismissed from consideration because substitutes such as scrap mica and talc, which are both abundant and cheap, may be freely used therefor. These same two minerals compete even now with graphite in lubricants, and for most lubricating purposes ordinary oils and greases can be used almost as well. For steam packings, use is made of asbestos, rubber, and other materials with suitable lubricants, and for boiler compounds graphite now competes on terms of practical equality with a rather large variety of chemical preparations. In pigments and paints, lampblack gives as good or better color than graphite, and various combinations are made up that duplicate any of the other useful properties of graphite as a paint material.

Numerous other preparations besides graphite are on the market for use in foundry facings, now by far the largest application of graphite. With the possible exception of certain silica-molasses facings most of these other preparations have a much more limited use. Without in any way seeming to detract from the excellent reputation of the various graphite facings, it can be admitted that, if need arose, foundries could wholly dispense with the use of graphite without injuring the quality of their castings or very substantially increasing their costs.

For most of these uses even the size of the particle is not of prime importance. It should be remembered that much of the finely ground crystalline graphite such as that used in foundry facings, to take only one example, is essentially a by-product of the production of crystalline flake for making crucibles or refractories. For most of such uses natural amorphous or artificially manufactured graphite would serve quite as well.

The statement has frequently been made that no satisfactory substitute for crystalline graphite in the manufacture of crucibles has been found. It has even been asserted more specifically that Ceylon plumbago is essential to the effective operation of munition factories even during a war emergency when price is not the major consideration. The experience of Germany during the Great War bears upon this question.

Normally Germany imported about 70 per cent of its graphite supply; part of this amount, however, was used for producing items which - like stove polish, paints, and pencils -- were not needed in such great quantities in war time. A shortage of graphite developed, but home production was increased by introducing better methods at the Bavarian mines (most of which had been operated in a small way by lessees) and imports from Austria Hungary were maintained. Bavarian graphite was found suitable for making crucibles and replaced the lump material formerly imported from Ceylon and Siberia. One method for rendering the small Passau flake more refractory and thus better suited for crucible use was to press it into larger aggregates, similar to briquets. Arrangements were made for turning over all used crucibles to manufacturers, who extracted the graphite so that it could be used over again; for this purpose hydrofluoric acid was often employed. Graphite for lubricating purposes was obtained from kish collected at ferro-silicon furnaces iron and steel works, and caustic soda plants.

The various measures for increasing supplies of graphite were supplemented by measures for reducing consumption. The principal savings resulted from the adoption of methods of melting that did not require crucibles. The use of crucibles was restricted to the melting of pure aluminum, pure zinc, silver and gold, copper and copper alloys, hard solders, and special steels. Numerous graphite substitutes containing 39 to 77 per cent carbon were found satisfactory for foundry facings and other purposes, although since many of them contained as much as 1 per cent sulphur, they had to be used with caution. In May, 1917, a laboratory was added to the Graphit-Vermittlungstelle for the purpose of developing economies in the use of crucibles and fostering the use of substitutes for graphite. This agency, originally formed under the auspices

of the association of German iron foundries, was eventually given complete control of all graphite distribution and trade throughout the country.

In the United States and the Allied countries the shortage of graphite during the World War never became acute. Expansion in domestic output of both natural and artificial graphite more than sufficed to take care of increased American needs. The War Trade Board recommended the employment of 20 per cent and eventually 25 per cent of domestic flake in the manufacture of crucibles, but this was largely for the purpose of conserving shipping space and because of the length of time required for shipments to be made from Ceylon and Madagascar.[15]

15 - The increased production during the war, it should be noted, was accomplished only because of largely increased prices.

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GRAPHITE

PART IV - STATUS OF THE AMERICAN GRAPHITE INDUSTRY

BY

Paul M. Tyler

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

G R A P H I T E

PART IV. STATUS OF THE AMERICAN GRAPHITE INDUSTRY[1]

By Paul M. Tyler[2]

GENERAL STATEMENT

Graphite is an essential war mineral and for that reason has received much attention from the standpoint of national defense. During the World War the situation with respect to graphite supplies, though it never became acute, caused considerable concern. Even in peace time, however, graphite has aroused more interest than the actual size of the industry seems to warrant. Despite this wide interest the facts of the industry do not seem to be generally known. Often there has been an almost total disregard of the essential economic and technical problems involved. Long ago Bastin[3] made the following statement in a Government report: "To-day there are more abandoned mines and mills in the United States than the number in operation * * *. In the number of times some of these properties have changed hands in the course of a few years, there is a record of misrepresentation and disappointment that can hardly be equaled in any other branch of mining, and many properties have been notoriously associated with stock manipulation of doubtful character."

According to one authority fully 600 graphite mines have failed in the United States, and the record shows that of all these properties scarcely a score have ever succeeded in operating for as much as a year or two. More than 40 mines have been worked at various times in the State of New York alone and at least that many in Alabama. Throughout the country, in fact, there are scarcely a dozen States in which efforts have not been made at some time to exploit graphite deposits. At the present time the list of mines that can be classed as at all active is reduced to less than ten. Failures inevitably occur in all kinds of mining, but the disheartening feature of the record of graphite mining in this country is that so few ventures have even paid back the money put into them and that none has been a conspicuous success.

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1 The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be made: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

2 Assistant to chief, economics branch, U. S. Bureau of Mines.

3 Bastin, E. S., Graphite: Mineral Resources of the United States, U. S. Geol. Survey, pt. 2, 1911.

Many persons seem to have a mistaken idea as to the actual size of the graphite industry. The total domestic consumption of graphite is so small that, were it merely a matter of tonnage, one moderately large company could easily supply all of it. In the aggregate the United States seldom uses more than 30,000 short tons of graphite annually. The total value of this consumption -- as obtained by adding the proceeds from sales by domestic mines to the foreign market value of imported material plus an allowance to cover the probable value of artificial graphite -- normally is only about one and a half million dollars. Even this rather small consumption is divided up among at least four fundamentally different kinds of natural and artificial graphite and a much larger number of real or fancied varieties. As a result of this condition the maximum potential sales from any one mine in the United States can scarcely amount to more than \$200,000 or \$300,000 annually. According to the testimony in the recent Tariff Hearings, 87 per cent of the domestic production of flake graphite is furnished by one company.[4] This company, certainly the most successful now operating, has expended more than \$2,000,000 in developing its mines and in the construction of milling equipment and the gross value of its annual sales as indicated by the total value of domestic production in recent years has been substantially less than \$200,000. Even were the margin between costs and selling prices very much greater than it actually is, the potential returns of new graphite mining ventures would not appear commensurate with the capital investment required.

More significant even than the facts concerning the present small tonnage requirements of specified grades is the question which confronts a prospective new producer as to whether his product can be sold at all. For certain uses and in certain localities it may be possible to market a small quantity of almost any graphitic material provided it is cheap enough, and provided, of course, it contains a fairly high percentage of carbon. But if the price demanded, even for powdered material, is more than 2 cents a pound, the problem of satisfying buyers as to quality becomes of paramount importance.

Under existing circumstances it is not possible for a graphite miner to expand his market except by engaging in the manufacture of finished products. That there is a wide spread between the price of crude graphite and the price of graphite products per pound is well known. A measure of this spread is indicated by a statement in the Tariff Hearings which pointed out the fact that one company whose sales amounted to \$1,500,000 annually paid out only \$15,000 annually in duties. Since the tariff rate on amorphous graphite under the act of 1922 was 10 per cent on the value of the material f. o. b. mine, it is evident that, even allowing for certain additional materials that enter into the finished articles, the average market value of graphite can be increased about 10-fold by manufacturing it into a varied line of finished products. Before entering these fields, however, the miner must build up a sales organization and conduct an active advertising campaign and otherwise popularize his brands before he can get consumers to take his products. Even the business of manufacturing paint pigment alone requires a great deal of sales effort and additional preparation beyond that connected with the purely mining needs of the business.

4 Brief of the Southwestern Consolidated Graphite Co. Tariff Hearings, Ways and Means Committee, 1929, p. 951.

PRODUCTION vs. CONSUMPTION

For some years previous to 1914 only about 15 per cent of the apparent consumption of natural graphite in the United States was supplied by domestic mines. In 1914 the proportion was 18 per cent. Even taking the industry as a whole and including both artificial graphite (exclusive of electrodes) and natural graphite, domestic sources contributed only 24.6 per cent of the apparent tonnage consumed in 1914. In 1926 the proportion derived from domestic sources was 50.5 per cent and in 1927, was 42.0 per cent, but this increase is attributable almost wholly to a larger output of artificial graphite. Considering natural graphite alone the proportions for the two recent years were 17.2 and 19.3 per cent, respectively -- only a little greater than the pre-war percentage.

Aside from the increased production of artificial graphite the leading changes in the industry have taken place in the import trade. The quantity and nature of the graphite produced by domestic mines have changed little, but the imports have been reduced substantially in quantity and to a much larger extent in value. As will be seen later the total tonnage of graphite consumed in the United States is about the same now as it was in the years immediately preceding the outbreak of the Great War in Europe. The outstanding difference is that a larger part of this consumption now consists of artificial and amorphous graphite and a much smaller part of it consists of Ceylon lump and expensive grades of flake. On the one hand the decline in the manufacture of crucibles has reduced the demand for the more expensive kinds of graphite, while on the other hand the consumption of the cheaper kinds of graphite has increased.

COMPARABILITY

No problem of the graphite industry is productive of more argument than the question of the relative merits of different kinds of graphite. An unbiased investigator soon finds that the conflicting views of different dealers and users are founded upon honest differences of opinion. In the manufacture of crucibles, for example, various mixtures of clay and graphite are employed and each mixture has to be handled differently. Experience has shown that one kind of clay can not be substituted for another without changing other factors in the process, and that the same is true of different kinds of graphite. Since it takes at least a year to work out the manufacturing technique to test the merits of a new mixture in actual melting service, it is not surprising that manufacturers are reluctant to depart from established practices.

The graphite industry, it should be remembered, is in many respects a complicated one. Attempts to state its problems as broad generalities tend to obscure the facts that there are three or four wholly different types of graphite with several grades of each type. Moreover graphites of nominally the same grade may not be interchangeable because of differences in purity and physical condition. Despite occasional attempts at standardization, the specifications for many grades are not well established, and even when established they are not always adhered to. A great deal depends upon the process of beneficiation and the care given to

maintaining quality or at least an even grade. Since one type of graphite can to a limited extent be substituted for another, the matter of careful grading is of extreme importance. The foundry-facing trade, for example, uses crystalline dust, low-grade flake, and amorphous or artificial graphite more or less interchangeably; at the same time all of these graphites compete not only with one another but with amorphous carbon and with mineral facings such as mica or silica. Since foundry facings, despite their wide variety, are not merely hit or miss mixtures, the miner or dealer who can furnish a dependable guarantee of quality has a great advantage in competition with a sporadic producer whose occasional shipments almost of necessity will be uneven in grade. In a statement presented to the Ways and Means Committee in February, 1919, 14 manufacturing jobbers engaged in the plumbago foundry facing business maintained that the Alabama dust carries grit, is harder to grind than the imported, and is not naturally as well fitted for foundry facings as the Ceylon graphite. Both Alabama and Pennsylvania graphite dust were said to be of inferior refractory quality because of their large silica and mica content which fuses under the pouring heat of the molten metal and prevents a smooth surface. These troubles are to some extent a matter of inadequate preparation for market. The same difficulty has contributed to the attitude of crucible manufacturers, many of whom objected to using domestic flake even as an admixture with Ceylon plumbago. The relatively rapid adoption of artificial graphite as compared with the consumption of competing types of natural graphite has to an important extent been due to the fact that the quality is under control and can be depended upon.

The successful introduction of the oil flotation process for cleaning graphite ores should result in better uniformity of domestic natural graphite, particularly of the flake variety, and may result in their displacing certain kinds of imported graphite. If at the same time domestic flake can be produced more cheaply it may react adversely upon the sales of domestic amorphous. In considering consumption trends and competitive conditions, therefore, it is necessary to keep clearly in mind that a number of different products differing widely in properties and price are all classed together as graphite.

That there is a great difference between Ceylon lump selling for almost 10 cents per pound and Rhode Island amorphous selling for under three-fourths of a cent is self-evident, but it is not so plain that domestic crystalline flake may range in price from $2\frac{1}{2}$ to 10 cents per pound as was the case in 1925, according to reports of producers to the Bureau of Mines. Part of this difference was explained, of course, by the fact that these prices were at the mines and hence did not include transportation to consuming markets; but one of the main causes for the wide range in value was the greatly different degree to which the products were prepared for market. This situation is reflected by the divergent reports of market conditions that are received from different dealers but more particularly from domestic miners. Even in the same district one operator finds business good while at the same time another fails to move his product. Among the imports there are similar differences, although the tendency is now for graphite to be more highly refined before being sent to this country. Ceylon graphite has always been carefully prepared, but much Madagascar and Canadian flake, for example, formerly had to be cleaned in this country before it was fit to be used. Under these circumstances an unknown percentage of the total tonnage reported in the statistics as graphite actually represented impurities,

and consequently the yield of usable graphite was substantially less than is indicated by the statistics.

TOTAL CONSUMPTION

Considering the graphite industry as a whole the situation is illustrated in the following table:

Consumption of Graphite in the United States, 1914 - 1927

Year	Domestic sales a/	Add imports	Available for consumption	Less exports	Apparent consumption	Percentage domestic
1914	6,660	21,990	28,650	1,960	26,690	25.0
1915	7,260	23,075	30,335	524	29,811	24.4
1916	12,287	42,930	55,217	798	54,419	22.6
1917	18,830	42,577	61,407	2,573	58,834	32.0
1918	17,582	19,498	37,080	954	36,126	48.7
1919	11,504	26,626	38,130	629	37,501	30.7
1920	13,210	21,096	34,306	607	33,699	39.2
1921	5,381	8,183	13,564	921	12,643	42.6
1922	9,641	12,488	22,129	569	21,560	44.7
1923	19,419	19,434	38,853	908	37,945	51.2
1924	10,464	16,375	26,839	1,022	25,817	40.5
1925	10,733	17,768	28,501	945	27,556	38.9
1926	16,052	16,116	32,168	405	31,763	50.5
1927	11,326	17,452	28,778	1,819	26,959	42.0

a/ Including both artificial (except electrodes) and natural graphite.

The foregoing table brings out in a general way the dependence of the United States on foreign sources of supply. To a certain extent also it shows that this dependence is less serious now than formerly, although these altered circumstances would be more apparent if it were possible to show figures for pre-war years. Unfortunately, however, the production figures for artificial graphite prior to 1915 related both to electrodes and to powdered or bulk graphite, whereas more recently such figures relate only to powdered graphite. It would seem, however, that the output of artificial powdered graphite for most years prior to 1914 was relatively small; as the imports in 1912 and 1913 were much larger than they were in 1914, it follows that previous to the outbreak of the war in Europe the proportion of graphite supplies derived from domestic sources amounted to substantially less than one-seventh of the consumption.

As has been previously noted, the leading causes of this increased independence of foreign supplies are (1) a decreased demand for crucible graphite and (2) an increased output of artificial graphite. Although imports of crystalline and amorphous graphite have been recorded separately in the trade statistics only since the Tariff Act of 1922, it is possible to estimate fairly closely the imports for previous years by noting the country of origin. Ceylon graphite, of course, is crystalline, as are also imports from Madagascar and Canada. Since a little Madagascar and Ceylon material has been imported by way

of Great Britain or France, imports from these and from a few other secondary sources have been classified as crystalline, while imports from Japan, Chosen, Mexico, Italy, Austra, and so on, are considered to be amorphous.

CONSUMPTION OF CRYSTALLINE GRAPHITE

The following table, with figures estimated as indicated, shows a diminution in the consumption of crystalline graphite from the 20,000 short tons immediately before the war to approximately 10,000 tons after the war. The drastic curtailment both in domestic sales and imports in 1921 and 1922, following the relative large imports of previous years, indicates the absorption of surplus stocks.

Consumption and sources of supply of crystalline graphite in the United States, in short tons, 1912 - 1927

Year	Domestic sales	Imports a/	Apparent consumption	Percentage domestic
1912	1,772	19,508	21,280	8.3
1913	2,532	19,288	21,820	11.6
1914	2,610	11,083	13,693	19.1
1915	3,537	18,995	22,532	15.7
1916	5,466	32,160	37,626	14.5
1917	5,292	32,462	37,754	14.0
1918	6,431	13,313	19,744	32.6
1919	4,043	20,972	25,015	16.2
1920	4,816	16,115	20,931	23.0
1921	595	4,085	4,680	12.7
1922	925	8,445	9,370	9.9
1923	1,982	8,182	10,164	19.5
1924	900	5,439	6,339	14.2
1925	1,129	8,558	9,687	11.7
1926	2,495	7,866	10,361	24.1
1927	2,612	7,893	10,505	24.9

a/ Imports for consumption; figures for crystalline graphite reported separately by Bureau of Foreign and Domestic Commerce only since 1922. Data for former years estimated on basis of countries of origin.

CONSUMPTION OF AMORPHOUS AND ARTIFICIAL GRAPHITE

The consumption of amorphous graphite and of powdered artificial graphite may be considered together, as the two materials compete quite freely with one another in many different uses. It should be noted, however, that recently a large percentage of the output of artificial graphite has been used in the manufacture of dry cells, competing in this application with flake rather than with amorphous. Because of the altered policy of the Acheson Graphite Co. in reporting

its statistics before and after 1915, it is impossible to give figures which will show the full extent of the expansion in the use of these two qualities of graphite. Moreover, while it is known that a large part of the exports of "unmanufactured graphite" as reported by the Bureau of Foreign and Domestic Commerce consists of amorphous or artificial graphite, minor amounts of crystalline graphite have been exported, particularly during the war years, but since there is no means of estimating the quantity of crystalline graphite so included, all of the exports are herein considered as being either amorphous or artificial graphite.

In general it may be said that the reduction of 10,000 tons in the consumption of crystalline graphite has been balanced as regards quantity by an increase of approximately the same amount in the consumption of amorphous and artificial graphite. Much of this increase is attributable to the great expansion in production of artificial graphite, although imports, most of which come from American-owned mines in Mexico, have also increased. Domestic-mine production of amorphous graphite prospered more or less during the war, but it subsequently appears to have suffered from the competition both of artificial graphite and of the dust and second-grade flake produced more or less as a by-product of mines that produce crystalline flake. Various kinds of amorphous carbon, notably anthracite coal, and other substitutes have also been factors affecting the consumption of natural graphite.

Consumption and sources of supply of amorphous and artificial graphite in the United States, in short tons, 1912-1927

Year	Domestic Production Natural amorphous(a)	Domestic Production Artificial (b)	Total	Add Imports	Apparent supply	Less exports (c)	Apparent consumption	Percentage domestic
1912	2,063	(d)	(d)	6,023	(d)	2,320	(d)	(d)
1913	2,243	(d)	(d)	9,501	(d)	2,692	(d)	(d)
1914	1,725	(d)	(d)	10,907	(d)	1,960	(d)	(d)
1915	1,181	2,542	3,723	4,080	7,803	529	7,274	51.2
1916	2,622	4,199	6,821	10,770	17,591	798	16,793	40.6
1917	8,301	5,237	13,538	10,115	23,653	2,573	21,080	64.2
1918	6,560	4,591	11,151	6,185	17,336	954	16,382	68.1
1919	3,379	4,082	7,461	5,654	13,115	629	12,486	59.8
1920	4,694	3,700	8,394	4,981	13,375	607	12,768	65.7
1921	1,842	2,944	4,786	4,098	8,884	921	7,963	60.1
1922	2,200	6,516	8,716	4,043	12,759	569	12,190	71.5
1923	4,056	13,381	17,437	11,252	28,689	908	27,781	62.8
1924	4,071	5,493	9,564	10,936	20,500	1,022	19,478	49.1
1925	3,536	6,068	9,604	9,210	18,814	945	17,869	53.7
1926	2,975	10,582	13,557	8,250	21,807	405	21,402	63.3
1927	2,595	6,129	8,724	9,559	18,283	400(e)	17,883	48.8

(a) Sales.
(b) Powdered graphite (exclusive of electrodes).
(c) Probably includes some crystalline flake in some years.
(d) Data not available previous to 1915. Artificial graphite was included with electrodes in published statistics.
(e) Estimated. Under new classification exports of "graphite and manufactures except crucibles," in 1927 were 1,819 short tons valued at $435,997. Since exports of manufactures of graphite alone had a value of $463,084 in 1926, it is assumed that exports of unmanufactured graphite were probably no larger than in the previous year.

COMPETITIVE CONDITIONS

The competitive ability of the domestic graphite industry depends to an important degree upon the quality of its products. Established reputation and buying habits induced by the long dependence of consumers upon foreign sources of supply afford definite problems to be met by an expanding domestic industry, but certainly outside of the crucible field, they do not constitute insurmountable handicaps to a domestic producer who can guarantee regular delivery and good uniform quality.

Competition in the domestic graphite market is becoming more and more a matter of price. This is certainly true with respect to competition between products of the same variety and purity; and in the case of many consuming outlets, it is true also that products of different grades compete with one another largely on a price basis. This latter statement, of course, does not mean always that they are interchangeable pound for pound at the same price, but it does mean that consumers as a rule are inclined to buy any of several varieties of graphite which may be offered to them and pay according to the relative service rendered. Formerly they were less ready to do this, preferring always to buy Ceylon graphite or some other variety with a well-established reputation. Sometimes consumers are willing to buy an impure product and refine it themselves, but the tendency is to place the burden of preparing graphite for final consumption upon the producer, where it rightfully belongs. Moreover, the employment of perfected process of oil flotation makes it easier now to produce a finished product at the mines.

Previous to the World War, Ceylon lump and even chip commanded a considerably higher price per pound than flake graphite, but in later years the differential has been greatly reduced. Throughout the world Madagascar flake has displaced Ceylon grades to a greater or less extent even in the manufacture of crucibles, and in consequence Ceylon producers have been forced to reduce their prices. In 1913, first-grade Ceylon chip was quoted in New York at 7½ to 10 cents per pound, whereas Madagascar, Canadian, or domestic flake could be bought for 5 or 6 cents per pound, indicating an average premium of at least 3 cents per pound in favor of this medium size of vein graphite. In 1928, on the other hand, Ceylon chip and the various grades of high-grade flake were all quoted in New York nominally at the same price, 6½ to 7½ cents per pound. Actually domestic and Madagascar flake were still somewhat cheaper, but the difference was very slight as compared with former times. The premium on Ceylon plumbago which, because of the enlarged demand for crucibles increased greatly during the war, has subsequently tended to disappear. Even for Ceylon graphite the differentials between the different grades have been much reduced as may be seen from the following table:

Quotations of Ceylon graphite per pound in New York for selected years

	1913	1917	1928
Lump (1st grade)	9½-11	28-32	7½-9½
Chip (1st grade)	7½-10	20-23	6½-7½
Dust (1st grade)	4 - 5½	11-13	3½-5½

Almost regardless of price, Ceylon lump and chip will continue to be used as raw material for crucible making. The only known domestic mine which has produced material physically similar to Ceylon graphite is situated at Dillon, Mont. Except at war-time prices, the Crystal Graphite Co. which operates this mine has never yet demonstrated an ability to compete in eastern markets with Ceylon producers. Development has continued and the mine has been opened up so that it can be worked to better advantage; a small mill has been built to prepare the products for market, but the sales in recent years have been confined to western consumers and future plans appear to be based largely upon the western market which is not large. These facts indicate that vein graphite can not now be produced in Montana and delivered in eastern consuming centers at the prices of comparable Ceylon grades. Transportation is one factor; it costs more to haul the Montana product to the Atlantic Seaboard by rail than it does to ship the imported material halfway around the world by water. Differences in wages tend further to handicap the American producer, who must meet the western mining camp scale of wages although the deposit does not afford much opportunity for compensating economies that will reduce unit cost by use of large-scale methods.

With the decline in the demand for crucible grades of graphite, the market for flake has improved -- at least relatively. Madagascar has established itself in many markets, displacing Ceylon plumbago both in the United States and abroad. Madagascar flake, since it has not the established reputation which is still accorded to the better grades from Ceylon, must sell on an equal footing with comparable domestic flake. It has been stated that Madagascar flake is a little thicker than that produced in Alabama and the new guarantees as to uniformity provided by the Government seal have been used as sales arguments, but although imported and domestic flake may not be interchangeable, they do sell at very nearly the same price. Under these circumstances the relative cost of the finished product delivered in the principal consuming centers of the United States is the determining factor in competition. Despite the relatively high value of the product, transportation is an important item, and for this reason Madagascar flake is consumed mainly along the seaboard. Canadian graphite is most strongly entrenched in the neighborhood of the Great Lakes. Alabama graphite mines, on the other hand, are not well situated with respect to consuming markets. The freight rate to New York is $16.10 per short ton, whereas the rate from Madagascar, including transhipment at Havre, is $16 (it has been as low as $8.67), and the rate from Ceylon to Atlantic seaboard points of the United States ranged in recent years from $5.50 (1928) to $8.75 (1925). To Pittsburgh the rate from Alabama points is $16.50, while from Canadian mines it is only $9.50. The Texas mines likewise are far from the main consuming centers, hence their product formerly had to be sold to a large extent locally, mostly in the southern part of the Mississippi Valley. California production (used mainly for foundry facings) also has to be sold in nearby territory. Only one mine in the United States, that at Annandale, N. J., 55 miles from New York, is situated near the great eastern markets, and in 1928 this property had not started to produce on a commercial scale.

The freight rates (1928) from the leading domestic mines to certain consuming districts are given in the following table:

Freight rates (carloads) from domestic mines per short ton in 1928

Destination	From Alabama (Hollins)	From Texas
New York	$16.10	$ 9.72 (a)
Chicago	16.30	10.35
Madison	- -	10.35
Bethlehem, Pa.	16.10	- -
Philadelphia	14.90	- -
Pittsburgh	16.50	12.95
Baltimore	13.30	- -
Ohio points	10.70	11.65 to 12.00

(a) By water.

In addition to the transportation handicap resulting from the fact that the main consuming centers are closer to the Atlantic seaboard or the Canadian boundary than they are to many of the leading domestic producing centers, domestic deposits are of lower grade than many of those found abroad. Regardless of labor conditions and relative wages the domestic deposits are more costly to work because the yield of flake per ton of rock mined is almost invariably less and generally very much less than it is abroad. In Madagascar there are deposits that yield 30 and even 50 per cent. In Alabama the yield amounts to scarcely $2\frac{1}{2}$ per cent; in Texas and Pennsylvania it is only 3 to 4 per cent, whereas in Canada the rock at the Black Donald mine contains an average of 65 per cent, and elsewhere in Canada graphite mines have been opened in rock containing 25 per cent graphitic carbon. The Canadian graphite industry, therefore, seems to be in a peculiarly advantageous position. The deposits are extensive and not far distant from the large markets of the United States, power is cheap, and cost of production is so low that Canadian flake should be laid down in New York, Pittsburgh, London, Liverpool, Hamburg, or Bremen, cheaper than that produced anywhere else in the world. In the past the Canadian industry, in common with that of the United States, has been held back by inefficient methods of beneficiation. The adoption of oil flotation, however, promises to do away largely with this difficulty. At the Black Donald property coarse flake containing 88 to 99 per cent graphite has been regularly produced; the No. 2 (medium) flake contained 92.5 per cent graphite while the No. 3 (fine) contained 95.5 per cent graphitic carbon. About 20 per cent of the total product is used for ground stocks. The graphitic carbon content of which ranges from 74 to 92 per cent. Some portion of the Canadian graphite which is finely crystalline is commercially known as "amorphous" and is imported under that designation.

A factor which tends to strengthen the competitive status of both domestic and Canadian graphite mining is the increase in the demand for dust and the smaller sizes of flake which has resulted from the rapidly growing consumption of graphite for foundry facings, paints, dry batteries, and lubricants. Formerly, when the demand was principally for No. 1 flake suitable for making crucibles, these joint products often had to be thrown away. According to the leading domestic producer, however, it is "absolutely impossible for any plant to survive on the sale of low-grade products alone."[5]

Some of the amorphous graphite produced in the United States is low in carbon, and none of it sells for high prices. Graphite of this sort is mined chiefly in Rhode Island, Nevada, and Michigan, and its principal recommendation is that it is cheaper than any suitable material obtainable from abroad. Some of it is worth only $5 to $10 per ton at the mines. As even the New York price of crude amorphous graphite is usually under $20, it is clear that domestic mines situated near consuming centers have a tremendous advantage with respect to supplies that have to be imported from any considerable distance. Since much of the amorphous graphites must be pulverized before entering into final consumption, the principal competition encountered by domestic producers of this kind of material comes from the low-grade dust produced in the refining of crystalline flake. For many purposes, on the other hand (notably for the manufacture of pencils and for lubricants and to a lesser extent for foundry facings), quality is of great importance even for amorphous graphite. This feature has been emphasized by operators of high-grade graphite mines in Sonora, with the result that Mexico now furnishes a larger and larger percentage of the imports of amorphous graphite, and products made therefrom -- especially pencil graphites -- are exported in substantial quantity. During the first half of 1928 almost 4,000 tons of graphite were imported from Mexico -- a considerable increase as compared with 6,283 tons imported from the same source in 1927. The total domestic production of amorphous graphite, most of it consisting of extremely low-grade material, is much smaller, amounting in 1927 to only 2,595 short tons. The United States has an abundance of amorphous graphite, deposits being found in many States and in various parts of the country. Both Mexico and Chosen, however, have larger and better deposits and, despite their greater distance from American markets, have captured much of the business where quality counts.

Under the Tariff Act of 1922 a duty of 10 per cent ad valorem was placed on amorphous graphite, but nevertheless imports are substantially larger and domestic production is smaller than formerly when all graphite was on the free list. Even as compared with pre-war production the domestic output has increased only slightly. The use of artificial graphite, on the other hand, has expanded greatly, as it is cheaper than natural graphite of equal purity. Moreover, the larger supply of low-grade crystalline graphite obtainable at competitive prices has further contributed to discourage the working of amorphous graphite mines in the United States.

The freight rate from Sonora to Saginaw, Mich., where the bulk of the Mexican material is prepared for market, is $12.50 a short ton, and, based on the average declared value of $8.57 in 1927, the 10 per cent duty adds $0.86 per short ton. The ocean freight rate from Japan or Chosen to the Atlantic seaboard has averaged in recent years about $8.95 per short ton.

5 Burrage, Robert, Brief presented in Tariff Hearings: Ways and Means Committee, January, 1929, p. 952.

COSTS OF PRODUCTION

For the period 1918 to 1919, domestic operating costs, as ascertained by the Bureau of Mines, ranged from 6 to 14 cents per pound, averaging about 10 cents per pound of No. 1 flake.[6] In ascertaining these costs the No. 1 flake was charged with the entire operating expense and then credited with the income received from the sale of No. 2 flake and dust, which were considered as by-products. No allowance was made for depletion and depreciation. Such allowances would add perhaps 1 or 2 cents per pound. These costs, of course, represent the peak of war-time conditions and are not representative of the industry in later years. Although most of the Alabama plants have been closed, the following figures, said to represent the cost and market prices of crucible grades only, are of interest as indicating the advances that took place during the progress of the Great War.[7]

Year	Cost per pound, cents	Market price per pound, cents
1914	4	6-7
1915	5	7-8
1916	6	10-13
1917	8	10-20
1918	10	11-20

At the Tariff Hearings in 1919 and again in 1921 considerable evidence was submitted to show that the costs of production of domestic graphite were substantially higher than they were in foreign producing countries. As shown by cost statements submitted in confidence to the Bureau of Mines, this condition has been modified somewhat by improved methods of beneficiation and more economical mining, but it may be noted that on Texas graphite the freight and grinding costs alone amount to $1\frac{1}{2}$ cents per pound of product, and on Alabama shipments the freight equals 0.5 cent; these costs must be considered in view of the close margins on treating ore yielding less than 25 pounds of product per ton of rock milled.

The best available evidence as to foreign costs can be obtained from a study of the invoice values of graphite imported from the leading producing countries. These figures include profit and certain handling charges in the countries of origin but do not include ocean freight or handling charges and commissions in the United States. During the calendar year 1927 the declared value of flake graphite imported direct from Madagascar was 3.84 cents per pound, whereas the imports from Madagascar graphite by way of France were valued at 5.23 cents per pound as compared with 6.74 cents per pound from Canada. The average of all kinds of imported crystalline flake is 5.25 cents per pound.

6 Dub, George D., Preparation of Crucible Graphite: War Minerals Investigation, Series, No. 3, Bureau of Mines, 1918, p. 22.

7 Testimony of A. B. Conklin, Secretary of the Graphite Producers' Association of Alabama, before Ways and Means Committee, Sept. 26-7, 1919.

As estimated by the Compagnie Générale de Madagascar in January, 1927, the cost of Madagascar graphite, 85 per cent grade, f.o.b. Tamatave, was 1,450 francs for the Antsirabe region, 1,750 francs for the Anivorano region, and 975 francs for the Vatomandry region. However, the British-owned Maskar Co. which dominates the local market, estimated the costs for the Antsirabe district at only 1,600 francs as compared with 1,900 francs per metric ton which it was paying for 85 per cent material laid down at Tamatave warehouse.[8] With the franc stabilized at 3.9 cents, a price of 1,900 francs per metric ton is equivalent to 3.38 cents per pound as compared with the price of 3.84 cents indicated by the American import statistics as the average f. o. b. value, a difference of a trifle more than one-fourth of a cent per pound.

LIST OF GRAPHITE MINES AND PRODUCERS

Alabama:
- Chilton County:
  - Mountain Creek:
    - Bama Graphite Mines (Montgomery, Ala.)
- Clay County:
  - Ashland:
    - Alabama-Quenelda Graphite Corp. (Birmingham, Ala.)
    - National Flake Graphite Co. (Ashland, Ala.)
    - Superior Flake Graphite Co. (Chicago, Ill.)
- Coosa County:
  - Hollins:
    - Southwestern Consol. Graphite Co. (Boston, Mass. -- also operates in Texas.)
  - Sylacauga:
    - Diamond Graphite Co. (Alexander City, Ala.)

California:
- Los Angeles County:
  - La Crescenta:
    - Standard Graphite Corp. (Los Angeles, Calif.)
  - San Fernando:
    - The Los Angeles Graphite Co. (Los Angeles, Calif.)
  - Sagus:
    - California Graphite Co. (Los Angeles, Calif.)

Colorado:
- Gunnison County:
  - Quartz:
    - Graphite Corp. (Cleveland, Ohio.)

Idaho:
- Blaine County:
  - Ketchum:
    - Griffith & Hampton (Ketchum, Idaho.)

8 Mining Journal (London), "Graphite production of Madagascar in 1926": May 28, 1927, p. 465.

Michigan:
    Baraga County:
        L'Anse:
            Detroit Graphite Co. (Detroit, Mich.)

Montana:
    Beaverhead County:
        Dell:
            National Carbon Co., Lessee (New York City.)
        Dillon:
            Crystal Graphite Co.

Nevada:
    Ormsby County:
        Carson:
            Carson Black Co. (Oakland, Calif.)

New Jersey:
    Hunterdon County:
        Annandale:
            Annandale Graphite Corp. (Philadelphia, Pa.)

New York:
    Niagara County:
        Niagara Falls and Buffalo: (Artificial graphite)
            Acheson Graphite Co. (Niagara Falls, N. Y.)
    Warren County:
        Graphite and Ticonderoga:
            Joseph Dixon Crucible Co. (Jersey City, N. J.)
            (American Graphite Co.)

Oklahoma:
    Atoka County:
        Atoka:
            H. Y. McBride (Denver, Colo.)

Pennsylvania:
    Chester County:
        Byers and Chester Springs:
            T. D. Just Co. (Chester Springs, Pa.)
        Chester Springs:
            Harry A. Schmehl (Chester Springs, Pa.)
            (Not a producer - recovers graphite from old crucibles.)

Rhode Island:
    Providence County:
        Cranston:
            Graphite Mines Co. (Providence, R. I.)

Texas:
    Burnet County:
            Burnet: (Also in Ala.)
                Southwestern Consolidated Graphite Co. (Boston, Mass.)

Virginia:
    Roanoke County:
        T. O. McAdoo (Salem, Va.)

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INFORMATION CIRCULAR

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SOME PHASES OF COAL-MINE VENTILATION

BY

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SOME PHASES OF COAL-MINE VENTILATION[1]

By J. J. Forbes[2] and M. J. Ankeny[3]

IMPORTANCE OF VENTILATION

Ventilation is the most important safety factor in the production of coal because on it depends the safety, health, and efficiency of those who work underground. Without adequate and efficient means of supplying fresh air to the working faces, mining could not have advanced as it has. The extent of the workings, the increase in mechanization, and the increasing use of electricity are largely responsible for changes in methods of ventialtion.

Ventilation in the early days of coal mining was accomplished by means of a natural draft, created principally by a difference in the weights of columns of air between the intake and return openings. Later the furnace was introduced for the purpose of increasing the quantity of air in circulation. Mines continued to become larger, their output increased, men were employed in greater numbers, and more fresh coal surfaces were exposed. Coal beds near the surface, and practically free from explosive gas became exhausted, and deeper beds that liberated methane were opened as development advanced. Mechanical ventilation became necessary and was first accomplished by disc-type steam-driven fans. The old-fashioned steam-driven disc fan has generally been supplanted by the powerful electrically operated centrifugal fans in use at modern coal mines. In order to be secure against gas explosions and provide ample fresh air for persons employed in large coal mines, air must be conducted through the mine to all working places and to all other unsealed parts of the mine in sufficient quantities to dilute and render harmless all dangerous and noxious gases.

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1 The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be used: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

2 Supervising engineer, instruction section, safety division, U. S. Bureau of Mines.

3 First-aid miner, U. S. Bureau of Mines.

DANGERS OF INADEQUATE VENTILATION

Lack of adequate and efficient ventilation is the primary cause of gas ignitions in coal mines. Gas explosions occur only when there is an accumulation of gas mixed in suitable proportion with air. Explosive gas does not accumulate in properly ventilated mines. The secondary causes of gas ignitions are: (1) Improper installation, maintenance, or use of electrical equipment; (2) Use of open lights; (3) Use of nonpermissible explosive or the improper use of permissible explosives; (4) misuse of safety lamps; (5) matches and smoking; (6) mine fires. Until recently, open lights ranked second as a cause of explosions.

It is a matter of record[4] that from July 1, 1927, to June 30, 1928, there were 29 ignitions of explosive gas, costing the lives of 341 men in coal mines of the United States. Of these explosions, 14 were of electrical origin, 10 were caused by open lights, 3 were from explosives, and 2 were from unknown causes.

RESPONSIBILITY FOR PROPER VENTILATION

In order to have proper ventilation and promote health and safety, most coal-mining States have provided laws which require the operator to employ a competent and practical overseer, usually called the mine foreman or mine manager, for every mine. He is required by law to have a license or certificate, and such certificate is supposed to be sufficient evidence of his competency. The foreman is charged with the duty of keeping a careful watch over the ventilating apparatus, airways, entries, and all other parts of the mine to see that the provisions of the law are complied with. As the foreman is held legally responsible for the ventilation of the mine and as the maintenance of ventilation is the most important part of his job, he should be thoroughly conversant with the good and bad practices pertinent to coal-mine ventilation.

EFFECT OF OUTBURSTS OF GAS

Coal mines are usually classed as gassy and nongassy or as open-light mines and closed-light mines by State inspection departments or State laws. Some States permit individual mines to be divided into open-light sections and closed-light sections. As a rule a different set of laws from those applied to gassy mines is provided for so-called nongassy mines or nongassy sections of mines. Men who are not considered competent to serve as foremen in gassy mines are legally permitted to serve as foremen in so-called nongassy mines. More or less explosive gas is generated in all coal mines; in mines where gas has never been detected with a flame safety lamp it is not uncommon suddenly to encounter methane in the advance workings, sometimes when it is too late to make provision for taking care of it. Frequently there is a group of mines operating adjacent to each other in the same coal bed with the same thickness of overburden, yet one is classed as gassy and the other as nongassy. At the mines which are classed as nongassy or which are operating with open lights few if any precautions are taken

4 Harrington, D., Mine Explosions in the United States During the Fiscal Year Ending June 30, 1928: Information Circular 6085, Bureau of Mines, 1928, 4 pp.

to safeguard against a gas ignition in the event that an unexpected gas feeder is opened up or an accumulation of gas on top of a caved area is swept into the working places by a fall or by derangement of the ventilating current.

The fact that explosions often occur in mines that are not recognized or admitted to be gassy is borne out by records. The so-called nongassy mine which gives off small quantities of methane is more dangerous than the definitely gassy mine where precautions are usually taken against possible accumulations or ignitions of gas.

RECOMMENDATIONS OF THE BUREAU OF MINES

The Mine Safety Board of the United States Bureau of Mines has made the following decision (No. 1) relating to miners' lamps in coal mines[5]:

"(a) In all coal mines the portable lamps for illumination be permissible, portable, electric mine lamps; and also

"(b) In places where fire damp or black damp is liable to be encountered, a permissible magnetically-locked flame safety lamp for gas detection, or equivalent permissible device, be supplied to at least one experienced employee in each such place; and

"(c) Any employee before being supplied with a permissible flame safety lamp be examined by a competent official of the mine to assure the man's ability to detect gas; and

"(d) All coal mines whether classed as nongassy or gassy in any part, be supplied with magnetically-locked, permissible, flame safety lamps, properly maintained and in sufficient number for all inspection purposes."

All coal mines in which inflammable gas in excess of 0.05 per cent can be found by systematic search should be classed as gassy mines; however, in order to be practically secure against gas explosions, all coal mines should be managed, equipped, and ventilated according to the best safe practices recommended for gassy mines.

TYPES AND ARRANGEMENT OF FANS

In order to circulate air through the mine, a difference in pressure must be created between the intake and the return airways. In modern coal mines, this effect is produced by a fan preferably of the centrifugal type, which develops a much higher efficiency than the old and almost obsolete disc-type fan. Some fans are arranged to force air through the mine, others to exhaust it from the mine; during recent years, however, installations of the latter type have been more common. In the best practice the intake air should enter or downcast through

5 Mine Safety Board Recommendations of the Bureau of Mines on Certain Questions of Mine Safety: Information Circular 6091, 1928, 12 pp.

the hoist shaft or slope and main haulageroads, and should return or upcast through the air shaft. Fans should be constructed to permit quick reversal of the ventilating currents to meet emergencies due to fires or explosions, and the fan casings should have explosion-relief doors. Fans should be offset from the air shaft or air drift at least 25 feet from the projection of the nearest side so that they will not be damaged by an explosion. Fireproof construction of fans and casings should be required. When new fans are installed they should be of such size and efficiency as to produce the necessary volume of air against the natural resistance of the mine; much forethought should be given to the higher resistances likely to develop in the future life of the mine as the length of the airways increases and the volume requirements, the amount of exposed surface, and the number of men and animals increase[6].

Maintenance of Continuous Operation of Fans

Fans should be equipped with some form of warning device to attract the attention of the attendant if it has stopped from power failure or other cause. One method of doing this is to place a make-and-break contact device on the shaft of the fan to open and close an independent electric circuit with each revolution. In this circuit are red incandescent lamps at points where the attendant or outside foreman may observe them at all times. When the fan is running the lamps continue to blink, but when the fan stops, the lights discontinue "blinking" remaining either "on" or "off." Another device which is in use at some mines is an electric bell controlled by a water gauge and float; when the fan stops, the gauge float drops, permitting it to make an electric contact which closes an electric circuit and rings a bell; the bell continues to ring until the fan is restarted or receives attention. There are many other good methods of calling attention to fan stoppage.

Fans should be equipped with a continuous recording device which shows the water gauge of the mine. There should also be a similar device to record the speed of the fan. Foremen and others whose duty it is to help maintain the ventilation of the mine should always observe the water-gauge reading at the fan before making an inspection of the mine. When the recording water gauge shows any material change the mine officials should take extra precautions to ascertain the reason for the alteration.

Fans should be kept in continuous operation while there are men in the mine, and they should have an auxiliary drive, such as a steam engine or gasoline engine. If the fans are electrically driven, a separate source of power should be provided where the mine is known to give off explosive gas. When a fan stops and can not be immediately started, when it is necessary to stop the fan for any reason, or when any major changes are being made in the ventilating system, the men should be immediately removed from the mine.

If a fan is stopped purposely or by accident, all electric power underground should be immediately cut off unless the result of so doing would prevent

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6 The question of changing fans at certain periods during the development of a mine was discussed by W. A. Weldin and others of the Engineers' Society of Western Pennsylvania. See volume 39 of the Proceedings, 1923, pp. 113-114.

or seriously handicap escape of the men in the mine. An automatic device for opening main switches on electric circuits leading underground can be easily constructed in any mine machine shop as follows: Weld a piece of circular sheet metal with a one-eighth-inch hole in the center over one of the 6-inch openings of a 4 by 6 by 6 inch iron pipe to the 4-inch end of the T into a hole made in the air duct leading to the fan, and carefully cement the 4-inch end in place with the open 6-inch end of the T pointed downward. A circular $6\frac{1}{2}$-inch sheet-metal plate with a rubber gasket covering one side of it should be provided to fit over the open end of the T. This plate should be of such a weight that the negative pressure on the plate, created by the running of the fan, would be just sufficient to hold it in place over the mouth of the 6-inch hole of the T. Attach a piece of flexible wire to the center of this plate on the inside and pass it through the one-eighth-inch hole in the plate, welded on the top end of the T, to the lever of a low-voltage switch mounted directly above the T. When the fan stops, the negative air pressure on the bottom plate will be relieved; the plate will fall and in so doing pull the flexible wire which operates the low-voltage switch controlling a relay switch in the main power circuit.

Booster and Auxiliary Fans

The booster and auxiliary fans used for a number of years extensively and appropriately in metal mines have been installed in some coal mines. Most of these subsurface installations in coal mines are open to severe criticism because coal mining is attended by dangers from which metal mining, with rare exceptions, is free. The Mine Safety Board of the Bureau of Mines recommends in decision No. 4[7] that

> "Auxiliary fans or blowers should not be used in coal mines as a substitute for methods of regular and continuous coursing of the air to every face of the mine."

Quantity of Air for Ventilation

To conduct the air into and outby from the active working sections of the mine there should be at least three but preferably four, five, six, or more main parallel entries or slopes. The number of main entries required is determined largely by the proposed ultimate size of the mine, size of individual air courses, and quantity of air which is likely to be needed to ventilate the mine. In smooth-lined shafts and air courses the velocity in general should not exceed 1,800 feet per minute; in ordinary rough-ribbed airways it generally should not exceed 900 feet per minute; in main haulageways and manways the velocity should not exceed 600 feet per minute. On the other hand, the velocity of the air current of any one split should not ordinarily be less than 200 feet per minute, and where strata temperatures are high, the velocity required may be much greater than 200 feet per minute; the volume of a regular split should not be less than 10,000 cubic feet per minute in order to dilute and sweep away gases and dusts and provide the men with air that is within the limits of purity for health.

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7 See reference 5, Page 3

Decision No. 7[8] of the Mine Safety Board of the Bureau of Mines says:

"In the interest of safety, the Bureau of Mines recommends:

"1. That the main intake and main return air currents in mines be in separated shafts, slopes or drifts.

"2. That the main intake shaft lining be of fireproof construction, and that there be a minimum amount of inflammable material in or adjacent to the shaft."

STOPPINGS

Stoppings in breakthroughs between the main intake and return entries or shafts should be substantially built of fireproof material, and they should be "hitched" into ribs and floor composed of soft material. They should be carefully plastered and the ribs, floor, and top in the immediate vicinity of the stopping should also be plastered to prevent leakage. Main entry stoppings should be built of brick, cement block, tile, concrete, or similar material which will make them tight, solid, and incombustible.

Stoppings in breakthroughs on butt or developing entries should be substantially built of fireproof material, preferably brick, cement block, tile, or concrete. Gob stoppings well plastered on one side or brick stoppings laid without mortar but carefully plastered can be made effective. The advantage of using brick stoppings for this purpose is that the brick can be reclaimed on the retreat and used again. Under no circumstances should unplastered gob stoppings or board stoppings be used permanently in developing entries.

Overcasts should be substantial, fireproof, and leakproof. The theory that flimsy overcasts may help to check an explosion by blowing out and relieving the pressure should be given no consideration. The roof above and leading to and from the overcast should be so excavated as to leave an easy curve or incline overhead to create the least possible resistance to the flow of air, and the floor leading to and away from the floor of the overcast also should be a smooth incline.

DOORS

Main ventilating doors should be carefully installed with the frame, preferably of concrete, hitched well into the ribs. The doors should be of steel; if made of timber they should be of two thicknesses of matched boards, one thickness placed diagonally with fire-resistant canvas between. The doors should be so hung as to be self-closing by a suitable system of weights and pulleys or other equally effective means; if they are arranged for mechanical opening and closing, the arrangement should be positive as to keeping the door closed at ordinary times. All principal doors should be so placed that when one door is open, another which has the same effect upon the ventilating current shall be closed and remain closed to prevent any short-circuiting of the air current. No hook or similar device should be placed on any door to hold it open during temporary passage of cars nor should

8 See reference 5, Page 3

the propping open of doors be allowed. The mine should be planned and developed to eliminate doors or at least reduce the number of doors to a minimum.

AIR FLOW IN SPLITS

In order to get a sufficient flow of air in the longest splits in a mine, it is usually necessary to create an artificial resistance in one or more of the shorter splits. This result is accomplished by placing a regulator in the returns of the short splits, provided that such a position will not interfere with the operations in that split. A good type of regulator is a brick or concrete stopping with a rectangular hole that allows plenty of space for the passage of a man and has a horizontal slide to permit close regulation of the air current passing through.

In decision No. 9 the Mine Safety Board of the Bureau of Mines recommends in coal-mine ventilation the following specifications[9] as to unit quantity and quality of air:

"1. The quantity in cubic feet of pure intake air flowing per minute in any ventilating split shall be at least equal to 100 times the number of men in that split.

"2. The quantity of air entering each unsealed place shall be at least 200 cubic feet per minute and as much more as may be necessary to properly dilute and carry away inflammable or harmful gases which may be present.

"3. The air shall be made to circulate continuously to the face in every unsealed place into which an appreciable amount of methane enters.

"4. The air in any unsealed place shall be considered unfit for men if it shall be found to contain less than 19 per cent oxygen (dry basis), more than 1 per cent carbon dioxide, or a harmful amount of poisonous gas.

"5. If the air in any unsealed place, when sampled or tested in any part of that place not nearer than 4 feet from the face and 10 inches from the roof, shall be found to contain:

"(a) more than $1\frac{1}{2}$ per cent of inflammable gas, the place shall be considered to be in hazardous condition and require improved ventilation, and

"(b) if more than $2\frac{1}{2}$ per cent of inflammable gas is found, the place shall be considered dangerous and only men who have been officially designated to improve the ventilation and are properly protected shall remain in or enter said place.

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9 See reference 5, Page 3

"6. If the air in the split which ventilates any group of workings contains more than $1\frac{1}{2}$ per cent of inflammable gas, these workings shall be considered to be in a dangerous condition and only men who have been officially designated to improve the ventilation and are properly protected, shall remain in or enter said workings."

ACTUAL QUANTITY OF AIR TRAVERSING WORKINGS

Too much attention can not be given to the proper maintenance of doors, overcasts, regulators, and stoppings, for here air losses chiefly occur. A large quantity of air going into or out of a mine does not indicate that the mine is well ventilated. The foreman who is interested in the efficacy of his ventilating system should keep a careful check of the total quantity of air measured in the intake and return of each split against the total quantity of air at the last open crosscut in the split to determine the approximate percentage of air lost; he should also check the sum total of the air of the various splits against the total of the intake and the return of the entire mine to ascertain what part of the air entering or leaving the mine is put to actual use.

Only a few States specify where the measurement of the air currents shall be made. Generally, they permit the volume of air entering the mine to be measured at the foot of the shaft or in the entrance of a slope or drift. A good standard practice is to measure the total intake, the intake and return of each split, the air in last crosscut of each entry in a split, and the full return from the mine.

Tests have shown that in 16 Illinois coal mines an average of only 18.6 per cent of the entering air reached the last crosscuts nearest the faces because of leakage at doors, stoppings, overcasts, and through crevices in pillars and roof[10]. Good practice requires that at least 50 per cent of the air entering a mine shall reach the faces; it is possible to get as high as 80 to 85 per cent with good natural conditions and properly built doors, stoppings, and overcasts. It costs money to circulate air through a mine whether this air effectively reaches the working faces or not, and properly constructed stoppings, doors, and overcasts soon pay for themselves in the saving of power.

VENTILATING ROOMS AND ENTRIES

In order properly to ventilate developing rooms and pillar lines it is necessary to deflect the current of air into the rooms to be ventilated by means of light wooden or wooden and canvas curtains. Extreme care should be taken in the construction of deflectors of this type to prevent them from coming in contact with electric wires of any kind and creating a fire hazard. When gas is being given off or likely to be given off at the face, it may be necessary to erect canvas checks or doors in the entry at fairly frequent intervals; as an alternative the check canvas curtains or doors may be placed in the entry between a few of the rooms so as to thus deflect air into the rooms which can be held there by canvas curtains or light doors placed in the room necks. In addition, temporary

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10 Williams, R. Y., Mine Ventilation Stoppings, with Especial Reference to Mines in Illinois: Bull. 99, Bureau of Mines, 1915, 30 pp.

stoppings should be placed in all crosscuts between the first and second rooms and in the last and next to last rooms, excepting the crosscuts nearest the faces in these rooms; and in definitely gassy mines all room crosscuts but the one nearest the face should be stopped off.

Frequently, it is necessary to conduct the air from the last crosscut in rooms and entries to the face by means of a line brattice erected from the outby corner of the crosscut to a point 5 or 6 feet distant from the face. Room stoppings may be constructed of canvas securely nailed to posts or board frames supported by posts or by boards supported by posts. Line brattices should be constructed of a heavy canvas which preferably has been treated by a fireproofing material. Wherever practicable the brattice should be placed in a room or entry so that on the intake side of the canvas the area is smaller than the outlet side, thus providing a higher velocity of air to sweep the face free of generating gas. Where smooth roof and floor conditions exist, a line brattice may be stretched on posts, or preferably nailed to a substantial frame, but where very rough and irregular roof conditions prevail, holes should be bored in the roof at the high points, plugs inserted, and the canvas nailed to the plugs. It is important that temporary entry and room stoppings and line brattices be substantially built and made as air-tight as possible. Failure of these devices, due to accident, may quickly create a very serious explosion hazard at the face.

The use of small portable fans or blower fans in connection with canvas tubing or pipes for the purpose of ventilating headings, rooms, or narrow workings that are without crosscuts is not recommended. Where absolutely necessary to use these dangerous expedients, as in tunneling work, their installation as well as their operation should be very carefully safeguarded if gas explosions and fires are to be avoided.

VENTILATING WORKED-OUT AREAS

None of the several general methods of trying to ventilate worked-out areas can be considered even reasonably effective. One method is to cause the air to sweep along the edge of the breakline, so as to put a pressure on the caved area and dilute and carry away gas that may come off the caved area. Although this method generally provides fresh air for men working the pillars or stumps and ordinarily keeps the working places free from gas, the danger of a gas explosion is ever-present. In the event of an interruption of the ventilating current -- due to failure of the fan to operate, to a door left open, or to blocking the air current by a fall in the airway -- pressure on the caved area is relieved, thus permitting the explosive gas to come from the falls into the working places. A large fall back in the caved area may also force the gas out into the working places.

A method of ventilating caved areas, in use at many mines, is to pass some air over the caved regions into the return airways of adjacent sets of entries. In developing rooms occasional rooms are driven through the barrier pillar into the return of the adjacent set of entries and a regulator is placed in the room near the point where it connects with this return. When the retreat work has passed one of these rooms which has been holed through, the regulator can be opened to permit some air to flow through. With this method of ventilation, a current of air is

continually passing over the caved area and possibly carrying away explosive gas as it is being given off; however, this method can be used only in those mines in which the engineering "layout" is favorable to its use when sufficient air is supplied to the producing workings to permit losing a small proportion over the extracted areas. Air samples should be taken at these bleeders from time to time to determine the amount of air required to be passed through the regulators to keep percentage of explosive gas within safe limits.

Decision No. 6 of the Mine Safety Board of the Bureau of Mines recommends that:[11]

> "All entries, rooms, panels, or sections that can not be kept well ventilated throughout or can not be inspected regularly and thoroughly, or that are not being used for coursing the air, travel, haulage, or the extraction of coal, be sealed by strong fireproof stoppings."

When the difficulty of efficiently ventilating live advancing workings is considered, it seems beyond the realms of possibility to keep inaccessible caved abandoned regions even reasonably free of accumulations of methane; hence the safest procedure with worked areas is undoubtedly to seal them "by strong fireproof stoppings" as recommended by Bureau of Mines Safety Board Decision No. 10, quoted above.

REMOVAL OF GAS ACCUMULATIONS

Every mine should be kept free of standing gas in all parts except those sections which are effectively sealed. Any accumulation of standing gas should be removed as soon as possible after its discovery; until the gas has been removed no one should be allowed in the gassy portion of the mine except the men engaged in the removal of the gas under the direct supervision of the foreman. Before attempting to remove gas, all power circuits should be cut off in the portion of the mine affected and an ample quantity of air provided to keep the mixture, as it passes into the return, below the explosion point.

DEFINITIONS FOR VENTILATION PRACTICE

Decision No. 8 of the Mine Safety Board of the Bureau of Mines recommends[12] that in coal-mine ventilation the following definitions be used:

> "1. The term "intake air" and the term "return air" without qualifying adjectives shall be used only to define mechanical movement of the air respectively in an inward and outward direction with reference to the mine as a whole or to any one group of workings.

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11 See reference 5, Page 3
12 See reference 5, Page 3

"2. When health and safety are concerned, the term "pure intake air" shall mean:

"(a) Air which has not passed through or by any active workings, and (or)

"(b) Air which has not passed through or by any inactive workings, unless these are effectively sealed, and

"(c) Air which is free from poisonous gas and by analysis contains not less than 20 per cent oxygen (dry basis) and not over 0.05 per cent inflammable gas."

Records show that the improper installation and use of electrical equipment in other than pure intake air have become the most prolific source of ignition of gas and dust in initiating mine explosions. Of the 31 explosions investigated by the Bureau of Mines during the fiscal year ended June 30, 1928[13] 14 ignitions were caused by electricity and 282 men were killed in the resultant explosions. In order to use electricity with relative safety in other than pure intake air it is absolutely necessary to carry out certain specifications, among them the following:

1. The installation of trolley wires and the use of trolley locomotives should be permitted in pure intake air only.

2. Feed lines for mining machines should be covered with good insulation and supported on insulated hangers on the opposite side of the entry from the side on which men are permitted to travel.

3. The "nipping" or tramming of mining machines by sliding the nips along bare wires in other than pure intake air should not be permitted. Permissible junction boxes should be provided for the purpose of connecting machines and drill cables to the power feeder circuits. When junction boxes are provided at proper intervals along feed lines, tramming the mining machines from place to place by sliding a trolley nip along bare wires is unnecessary.

4. Animal haulage or permissible storage-battery locomotives only should be used for gathering coal in other than pure intake air.

5. All electrical equipment used in other than pure intake air fresh from the outside should be of the permissible type approved for safety by the U. S. Bureau of Mines. Such equipment should be inspected regularly and maintained in a permissible condition at all times.

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13 See reference 4, Page 2

INSPECTION FOR GAS

In order that the foreman may know that his mine is being maintained in a safe and healthy condition he sould give much thought to the matter of inspections. He should build up his organization by carefully selecting competent men to act as section foremen and firebosses. In addition to the legal requirements for section foremen and firebosses, these men should be capable and energetic men who have had considerable practical experience in gassy mines. They should be physically fit, particularly with respect to good eyesight, and should be proficient in the knowledge of limitations, use, and care of the flame safety lamp.

The fireboss, section foreman, and mine foreman should arrange the time of their daily visits to the working places while the men are in the mine so that the lapse of time between visits of mine officials is about evenly divided. Good mining practice demands that working places be inspected at frequent intervals and a proper record should be made of such inspections. A good rule for mine officials to adopt is to visit each working place at two-hour intervals, and more often if necessary. At some operations it is the practice to require the fireboss to perform other duties than those concerning safety, for which he is intended; in only too many mines the fireboss must build doors, measure deadwork or yardage, report condition of coal content of places, aid in timbering or in blasting coal, and so forth. The fireboss should not be required to perform duties other than looking after the safety of the men in his section of the mine. He should be required to leave a date mark and his initial at the face of every place examined and he should sign his report book at the end of each examination of his section and not at the end of the shift only.

There is too much laxity in the supervision of the night shift and during the periods when shifts are being changed. This is proved by the fact that many of the coal mines explosions of the recent past have occurred during these periods. From January 9 to June 20, 1928, there were six explosions which occurred during the night shift, during the period while the shift was being changed, or shortly before or after change of shifts. The time of ignition was as follows: 9:30 p.m., 7:45 a.m., 6:30 p.m., 7:45 p.m., 1:30 a.m., and 4:07 p.m. These six disasters killed 257 men.

MEASURING AIR VOLUME AND DETECTING GAS

The mine foreman should know how to measure air and should be conversant with the instruments for the detection of gas in order to perform properly the inspections necessary to determine whether the mine is being maintained in a safe condition.

The Anemometer

Many foremen when measuring air with the anemometer suspend it at arms length in the center of an entry and hold it in this stationary position until the measurement is completed. This method may be useful in making comparative measurements, but it is very likely to give incorrect information as to the quantity of air that is actually flowing past the point where the measurement is being taken. A good quick method which is accurate to about 10 or 20 per cent is to

traverse the area of the airway being measured. Select a place in the airway which is free from eddy currents and reasonably straight for a distance of about 100 feet both ways from the point at which the measurement is being taken. Start at the upper left-hand corner of the entry and move the anemometer down along the rib to the lower left-hand corner; move it slowly, steadily, and continuously from that point toward the right-hand rib for about one-fourth the width of the entry and then up toward the roof; then to the center and down to the floor; from this point over to about one-fourth the distance from the right-hand rib, and then up to the top, over to the right-hand rib, and down along the right rib to the floor. This procedure will give a good average reading of the velocity. The anemometer should be permitted to run at least one minute; longer time than this would tend to make the reading considerably more accurate. The correction factor of the anemometer should always be applied.

The Psychrometer

The use of the psychrometer is valuable in determining the amount of moisture being taken into and carried out of the mine by the ventilating current. It can be used to determine the percentage of saturation of the mine atmosphere at any point in the mine and also the temperature of the air in which men are working. Every foreman should be familiar with this instrument and with the meaning of the data obtainable by it.

The Flame Safety Lamp

The flame safety lamp is a comparatively simple device for quickly detecting the presence of explosive gas or a deficiency of oxygen in a mine atmosphere. It is the foreman's closest companion because many foremen carry it wherever they go while in the mine. It may be used to advantage also by section foremen, firebosses, shot firers, machine-runners, and all others whose duty it is to inspect or examine for explosive gas. A permissible lamp approved by the U. S. Bureau of Mines, properly assembled and tested out in the hands of a competent person who thoroughly understands the use of the lamp, is absolutely safe. During the last 20 years several hundred men have lost their lives because a few men have used flame safety lamps in ways that made the lamps dangerous. The bureau does not know of any explosion caused by a properly assembled, permissible flame safety lamp.

In 1917 an explosion in a coal mine in Colorado caused the death of 121 men. The firebosses of this mine were accustomed to report a mine free of gas if not more than a five-eighths-inch gas cap showed in their lamps. This practice in itself was dangerous because a five-eighths-inch cap meant that the atmosphere contained approximately 3 per cent of gas and that a slight change or derangement in ventilation might increase the gas to 5.5 per cent, making it explosive. The safety inspector, a trusted man, carried a key-locked lamp, the only one used in the mine. The lamp was found taken apart near his body and 32 matches were found in his clothing. Undoubtedly his lamp flame "went out;" then he took the lamp apart and tried to re-light it with a match. Instead he ignited the gas and thus caused a terrific explosion which killed every man and completely wrecked the mine.

It frequently happens that the foreman has a key-locked lamp, while the other underofficials are required to carry a permissible lamp. A mine foreman

should by all means have the right kind of lamp. He should not permit in the mine any flame safety lamps that are not magnetically locked. The Bureau of Mines does not approve any flame safety lamp unless it is equipped with internal igniter and magnetic lock.

The flame safety lamp has its uses and serves its purposes well, but it also has its limitations. The lowest percentage of gas which it will detect depends upon the observer. Some persons claim that they can detect as little gas as 1 per cent; others claim that $1\frac{1}{2}$ per cent is the least that can be detected. It is not unusual to see a fireboss or foreman come out of the mine to sign his report book, go to his locker or cupboard and get his spectacles, and then proceed to sign the book. It is impracticable for a man who can not see well enough to sign the report book without glasses to detect small quantities of gas without them, and practically no consideration is given to the fact that a man may be color-blind and unable to see the nonluminous flame of a gas cap. These facts are pointed out to show that there is a definite need for something to supplement the use of the flame safety lamp if not to supplant it.

HOW METHANE ACCUMULATES

Any conscientious mine foreman should know that when there is as little as $1\frac{1}{2}$ or 2 per cent of gas in any split in the mine he should recognize this as a dangerous condition and take steps to remedy it immediately by supplying more air to the split. It should interest him just as much to know that he had 1 per cent, 0.5 per cent, or 0.2 per cent. For example: 0.5 per cent of gas in a split through which 40,000 cubic feet of air per minute passes would mean that there would be 200 cubic feet of methane emitted in the split per minute; if the ventilation were disrupted for one hour there would be an accumulation of 12,000 cubic feet of methane which, if mixed with air in explosive proportions, would create 240,000 cubic feet of fire damp — enough to fill a 6 by 10 foot entry 4,000 feet long, or eight 200-foot rooms 30 feet wide by 5 feet high, full of explosive gas. It is obvious that some means of detecting the presence of methane in quantities lower than can be detected by a flame safety lamp would aid greatly in making coal mines safer from explosions.

In addition to the common practice of foremen and fire-bosses' examination with safety lamps, some system of daily or at least weekly sampling and analysis of return air currents should be in effect. Comparison of these analyses gives a daily or weekly check on the freedom from gas of those portions of the mine ventilated. For safety, the proportion of methane in return airways should be kept less than 0.5 per cent. However, the quantity of air necessary to keep the percentage of methane to 0.5 per cent in splits should not exceed 20,000 cubic feet of air per minute. When more than this quantity would be required to keep the methane content below 0.5 per cent, the work on that split should be slowed down or some other method devised to aid in removal of the methane.

DETERMINING THE METHANE IN RETURNS

The amount of methane in the return airways of splits may be determined by collecting air samples and analyzing them on the outside of the mine, or by

making determinations with the Burrell methane indicator inside the mine at the point where it is desired to know the methane content. This methane indicator is correct to 0.2 per cent and requires 5 to 8 minutes to make a determination. After a little practice it can be operated by anyone with ordinary intelligence. An electrical methane indicating device recently approved by the Bureau of Mines utilizes the effect due to burning gas on the surface of a filament which causes a change in resistance on a Wheatstone bridge, and an increase of temperature due to the combustion of the gas on the surface of the heated wire. This wire is connected with a set of fixed and variable resistances in an electrical circuit. Any change in the resistance of the wire causes a current to flow through a meter in the Wheatstone bridge circuit and the resultant movement over a scale graduated in per cent methane gives a direct indication of the amount of this gas present in the atmosphere to an accuracy of 0.2 per cent. Tests can be made with this device almost as quickly as with a flame safety lamp.

The mine foreman, section foreman, and firebosses before making their inspection of the mine should observe the water-gauge reading at the fan. An unusually low water gauge may indicate that the air has been short-circuited by a door left open, or that a fall has destroyed a stopping or overcast. An unusually high water gauge may indicate that a large fall in the airway is obstructing the flow of air; however, this procedure is not always reliable. They should make a test for gas in all working places, all places adjacent to working places, returns from splits, and tops of falls as far up into the gobs as possible. They should observe the direction of flow of air and should make frequent trips through return airways to see that they are free from falls.

CONCLUSIONS

The following conclusions may be drawn from the foregoing discussion:

1. Ventilation is the most important operation in the production of coal in connection with the safety, health, and efficiency of men employed.

2. There are very few coal mines, if any, where methane may not be found by systematic search.

3. Mines classed as nongassy, hence usually where proper precautions are not taken, are more hazardous than gassy mines where most of the known precautions are observed.

4. Permissible portable electric lamps and permissible flame safety lamps should be used in all coal mines.

5. A fan of proper size to meet the requirements of the mine should be installed on the surface.

6. Generally it is preferable to use an exhaust-type fan, so installed as to allow of reversing the direction of the air quickly if desired.

7. Auxiliary means of driving a fan should be provided, especially where it is known that any considerable quantity of methane is given off by any section of the mine.

8. If the fan stops, the men should be removed from the mine and generally all electric power should be cut off.

9. Fans should be equipped with an automatic warning device to attract the attention of the fan attendant or other responsible person if the fan stops.

10. Fans should also be equipped with a continuous recording device which shows either the water gauge or speed of the fan or both.

11. Booster or auxiliary fans should not be used for supplying air to working faces.

12. Preferably there should be three or more main entries to conduct the air into and out of the mine.

13. Stoppings in main and developing entries should be constructed of fireproof material and should be practically air-tight and substantially constructed.

14. The main intake and return air currents in mines should be in separate shafts, slopes, or drifts.

15. Overcasts should be substantially built of fireproof material; they should also be maintained free of leaks.

16. Ventilating doors should be carefully installed and maintained and careful provision should be taken against possible fires started by contact of electric wires with the timber or other inflammable material in doors or doorframes.

17. The quantity (cubic feet) of pure intake air flowing (per minute) in any ventilating split should be at least 100 times the number of men in that split, but the minimum quantity in regular splits should not in general be below 10,000 cubic feet per minute.

18. The air should be measured at the main intake, main return, the intake and return of each split, and the last open crosscut of each entry.

19. The air entering a split should be coursed to sweep the working faces by means of doors, stoppings, deflectors, and line brattice.

20. Small portable or blower fans should not be used except possibly for exploratory work in faults or in driving rock tunnels. Even in such cases they are dangerous, and their installation and use should be very carefully safeguarded.

21. Where possible, gob areas should be ventilated by passing air over the top of them.

Circular 6127 May, 1929

INFORMATION CIRCULAR

Department of Commerce -- Bureau of Mines

SURVEY OF CRACKING PLANTS

JANUARY 1, 1929

BY

G. R Hopkins

Circular No. 6127
May, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SURVEY OF CRACKING PLANTS, JANUARY 1, 1929 1/

By G. R. Hopkins 2/

According to reports received by the United States Bureau of Mines, Department of Commerce, as of January 1, 1929, there were 2,205 cracking units completed or being built in the United States, with a total daily charging capacity of 1,476,874 barrels. A similar survey of a year ago showed 2,334 units of 1,288,000 barrels total capacity. This indicates that despite a decrease in number of units, there was a material increase in capacity.

The year 1928 was, in general, an important one for the cracking branch of the industry. Gasoline consumption reached record levels, prices were higher than in 1927, and it is probable that in the late months of the year the cracking plants were operating very close to capacity.

Of the total capacity of 1,476,874 barrels for the completed plants and those under construction, 1,194,501 barrels, or 81 per cent, represents the capacity of the operating units, 147,923 barrels, or 10 per cent, was shut down, and 134,450 barrels, or 9 per cent, was being built. In comparison with the survey of January 1, 1928, this indicates an increase in capacity of the operating units of 18 per cent, a decrease in the shut-down capacity of 71 per cent, and an increase of 511 per cent in the capacity of the units under construction. On January 1, 1928, only 10 units of 22,000 barrels total capacity were under construction in only 5 States, whereas, on January 1, 1929, there were 71 units being built in 15 States to have a capacity of 134,450 barrels.

Texas easily retained its rank as the leading State from the standpoint of cracking equipment, both completed and under way. California recorded a capacity increase of 76 per cent in 1928 and displaced Indiana in second place. The East Chicago, Ind., district of the last-named State had the largest concentration of cracking equipment of any area of like size, with the possible exception of Houston, Tex.

The production of gasoline by the cracking process in 1928 amounted to 122,381,000 barrels as compared with 101,226,000 barrels in 1927, an increase of 21 per cent. Although the output of gasoline by straight-run methods and by the use of natural gasoline increased materially in 1928, the relative proportion of

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2 - Associate economic analyst, U. S. Bureau of Mines.

cracked gasoline to the total gasoline output rose from 30.7 per cent in 1927 to 32.4 per cent in 1928. The month of highest indicated cracking activity was November, when the proportion of cracked gasoline produced to the total was 34.3 per cent.

A number of refining districts produced over 40 per cent of their 1928 gasoline output by means of the cracking process. These were the East Coast district, embracing the refineries along the Atlantic seaboard, the Indiana-Illinois district, which comprises the central group of refineries, the Rocky Mountain and the Texas Gulf Coast districts. Although the output of cracked gasoline in California in 1928 was nearly double that of the previous year, the proportion to the total was only 10.8 per cent, less than one-third that of the majority of the other States.

In common with most other kinds of industrial equipment, the average size of cracking units has considerably increased. This fact was particularly well illustrated by this survey, which, in comparison with the 1928 survey, showed an increase in size of the average unit of from 552 barrels daily charging capacity to 670 barrels. Five years ago the majority of the new units projected were of 500 barrels daily charging capacity, but to-day it is not uncommon for units of 3,000 or 4,000 barrels capacity to be built.

Thirty-six different types of cracking processes are listed in this survey as compared with 31 in the survey of January 1, 1928. Since the majority of the types are used at only one refinery, there are only a comparatively few processes now being actively licensed. These are the Cross, Dubbs, Holmes-Manley, Jenkins, and Tube and Tank processes. All of these five types made gains in total charging capacity during 1928 and on January 1, 1929, had a combined capacity both built and under way of 948,121 barrels, equal to 64 per cent of the total. Although the tendency to dismantle the older shell type of Burton cracking stills was continued, there were a few cases in which the improved market conditions of 1928 apparently warranted further operation of some of these units which had been shut down.

The general desire on the part of the motorists for gasoline of high anti-knock qualities is reported to have given impetus to the active development of the vapor-phase system of cracking. This would seem to be substantiated by this survey which lists 2 vapor-phase units of 18,700 barrels capacity on January 1, 1929, as compared with 7 units of capacity 6,100 barrels on January 1, 1928.

RECAPITULATION BY DISTRICTS

District	Total units	Capacity, barrels per day			
		Operating	Shut down	Building	Total
East Coast........	273	192,300	44,400	39,200	275,900
Appalachian.......	77	40,500	3,417	5,700	49,617
Ind.,Ill.,Ky.,etc.	467	214,033	32,242	15,000	261,275
Okla.,Kans.,etc...	426	179,343	22,000	5,900	207,243
Texas.............	416	277,250	19,464	32,750	329,464
La. and Ark.......	77	84,600	750	23,000	108,350
Rocky Mountain....	394	63,775	22,650	1,600	88,025
California........	75	142,700	3,000	11,300	157,000
U. S. total..	2,205	1,194,501	147,923	134,450	1,476,874
Texas Gulf Coast..	354	248,500	16,014	6,000	270,514
La. Gulf Coast....	34	57,000	- -	15,000	72,000

RECAPITULATION BY STATES

State	Total units	Capacity, barrels per day			
		Operating	Shut down	Building	Total
Arkansas..........	24	11,200	750	- -	11,950
California........	75	142,700	3,000	11,300	157,000
Colorado..........	11	1,670	- -	- -	1,670
Georgia...........	6	2,000	- -	- -	2,000
Illinois..........	117	66,983	2,417	1,200	70,600
Indiana...........	260	119,800	24,800	9,000	153,600
Kansas............	146	62,600	8,000	2,500	73,100
Kentucky..........	37	8,000	- -	1,200	9,200
Louisiana.........	53	73,400	- -	23,000	96,400
Maryland..........	24	13,600	12,200	4,000	29,800
Massachusetts.....	18	27,500	- -	4,500	32,000
Michigan..........	2	- -	- -	3,600	3,600
Missouri..........	85	13,143	7,500	- -	20,643
Montana...........	1	- -	500	- -	500
New Jersey........	129	61,200	20,200	19,700	101,100
New York..........	11	11,800	- -	- -	11,800
Ohio..............	60	27,450	5,025	3,700	36,175
Oklahoma..........	195	103,600	6,500	3,400	113,500
Pennsylvania......	127	86,000	13,917	13,000	112,917
Rhode Island......	3	6,000	- -	- -	6,000
South Carolina....	8	10,000	- -	- -	10,000
Texas.............	416	277,250	19,464	32,750	329,464
Utah..............	4	7,500	2,000	- -	9,500
West Virginia.....	15	6,500	1,500	- -	8,000
Wyoming...........	378	54,605	20,150	1,600	76,355
U. S. total..	2,205	1,194,501	147,923	134,450	1,476,874

RECAPITULATION BY TYPE OF PROCESS

Type of process	January 1, 1929 Total units	January 1, 1929 Total capacity, barrels per day	January 1, 1928 Total units	January 1, 1928 Total capacity, barrels per day
Beacon coil........	4	4,000	(1/)	(1/)
Beggs..............	4	1,000	(1/)	(1/)
Black..............	12	12,000	(1/)	(1/)
Bruin..............	25	5,667	27	6,000
Buerger............	89	55,600	(1/)	(1/)
Burton.............	1,060	192,586	1,219	247,000
Carborundum........	1	3,000	(1/)	(1/)
Coast..............	80	14,000	(1/)	(1/)
Converter..........	1	3,000	(1/)	(1/)
Cross..............	160	232,250	154	211,000
De Florez..........	1	1,000	(1/)	(1/)
Digester...........	1	2,500	(1/)	(1/)
Doherty............	11	12,200	(1/)	(1/)
Donnelly...........	2	1,500	(1/)	(1/)
Dubbs..............	186	214,600	168	148,000
Fleming............	21	6,000	31	8,000
Floyd..............	3	300	(1/)	(1/)
General Pet. Corp. of Calif......	2	3,000	(1/)	(1/)
Gyro...............	11	8,200	(1/)	(1/)
Holmes-Manley......	154	232,107	150	207,000
Isom...............	100	100,000	102	102,000
Jenkins............	47	58,850	32	34,000
Knox...............	1	300	(1/)	(1/)
Koontz.............	4	4,500	(1/)	(1/)
Leamon.............	5	2,500	(1/)	(1/)
Lewis..............	38	14,000	(1/)	(1/)
Lientz.............	1	2,500	(1/)	(1/)
Link...............	20	53,000	(1/)	(1/)
Ormont.............	4	1,000	(1/)	(1/)
Richie.............	1	600	(1/)	(1/)
Setzler............	1	1,600	(1/)	(1/)
Slagter............	24	3,600	(1/)	(1/)
Sun Oil Co.........	3	16,000	(1/)	(1/)
Trumble............	2	1,000	(1/)	(1/)
Tube and Tank......	124	210,314	121	152,000
Winkler-Koch.......	2	2,600	(1/)	(1/)
Other..............	- -	- -	330	173,000
Total.......	2,205	1,476,874	2,334	1,288,000

1/ Included in "Other."

GASOLINE PRODUCTION BY CRACKING IN 1928

(Thousands of barrels of 42 U. S. gallons)

District	January	February	March	April	May	June	July
East coast..............	1,556	1,474	1,578	1,609	1,739	1,844	1,926
Appalachian.............	283	329	338	355	377	400	409
Ind., Ill., Ky., etc....	1,688	1,495	1,584	1,641	1,838	1,851	2,117
Oklahoma, Kansas, etc...	1,453	1,319	1,525	1,538	1,628	1,614	1,732
Texas...................	2,267	2,540	2,775	2,393	2,243	2,313	2,545
Louisiana and Arkansas..	538	546	647	613	594	581	728
Rocky Mountain..........	498	428	518	538	625	545	597
California..............	618	540	650	521	472	500	560
U. S. total........	8,901	8,671	9,615	9,208	9,516	9,648	10,614
Per cent of total gasoline production.......	31.8	32.3	32.8	31.4	30.9	31.4	32.2
Texas and Louisiana Gulf coasts...............	2,440	2,707	2,951	2,544	2,381	2,473	2,859

District	August	September	October	November	December	1928 Total	1927 Total
East coast..............	2,020	2,046	2,058	1,986	1,954	21,790	17,383
Appalachian.............	403	433	438	395	419	4,579	2,982
Ind., Ill., Ky., etc....	2,259	2,241	2,200	2,193	2,172	23,279	18,928
Oklahoma, Kansas, etc...	1,856	1,906	1,862	1,716	1,609	19,758	16,940
Texas...................	2,617	2,827	2,939	2,886	2,504	30,849	28,682
Louisiana and Arkansas..	711	820	753	802	808	8,141	6,865
Rocky Mountain..........	651	634	621	563	562	6,780	5,659
California..............	472	445	661	837	929	7,205	3,787
U. S. total........	10,989	11,352	11,532	11,378	10,957	122,381	101,226
Per cent of total gasoline production.......	32.2	33.7	33.5	34.3	32.3	32.4	30.6
Texas and Louisiana Gulf coasts...............	2,838	3,135	3,203	3,218	2,838	33,587	31,102

SURVEY OF CRACKING PLANTS, JANUARY 1, 1929

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	ARKANSAS				
Op.	Houston Oil Co. of Texas	Camden	2	2,000	Dubbs
Op.	Kettle Creek Refg.Co.	El Dorado	2	1,600	Dubbs
Op.	Lion Oil Refining Co.	El Dorado	15	4,000	Burton
SD.	Ouachita Valley Refg. Co.	El Dorado	1	750	Dubbs
Op.	Root Refining Co.	El Dorado	2	1,200	Dubbs
Op.	Simms Oil Co.	Smackover	2	2,400	Cross
			24	11,950	
	CALIFORNIA[1]				
SD.	General Pet.Corp.of Calif.	Los Angeles	2	3,000	Vapor Phase
Bldg.	Hercules Gasoline Co.	Los Angeles	1	1,500	Jenkins
Bldg.	Italo Pet.Corp. of America	Hynes	1	500	Trumble
Op.	Pan American Pet. Co.	Watson	12	12,000	Black
Op.	Petrol Gas Co.	Wilmington	1	1,200	Jenkins
Bldg.	Petrol Gas Co.	Wilmington	1	1,800	Jenkins
Op.	Richfield Oil Co. of Calif.	Hynes	4	12,500	Cross
Bldg.	Rio Grande Oil Co.	Vinvale	2	2,000	Jenkins
Op.	Shell Oil Co.	Dominguez	8	21,000	Dubbs
Op.	Shell Oil Co.	Martinez	1	3,000	Converter
Op.	Shell Oil Co.	Martinez	9	16,000	Dubbs
Op.	Shell Oil Co.	Watson	8	17,000	Dubbs
Op.	Standard Oil Co. of Calif.	El Segundo	8	20,000	Dubbs
Op.	Standard Oil Co. of Calif.	Richmond	1	2,500	Digester
Op.	Standard Oil Co. of Calif.	Richmond	7	17,500	Dubbs
Op.	The Texas Co.(Calif.)	Watson	2	5,000	Holmes-Manley
Bldg.	Union Oil Co. of Calif.	Wilmington	1	3,000	Carborundum
Op.	Union Oil Co. of Calif.	Wilmington	5	15,000	Cross
Bldg.	Union Oil Co. of Calif.	Wilmington	1	2,500	Lientz
			75	157,000	

1/ Data compiled by E. T. Knudsen, San Francisco Office of the U. S. Bureau of Mines.

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	COLORADO				
Op.	Continental Oil Co.	Florence	10	1,170	Burton
Op.	The Texas Co.	Craig	1	500	Holmes-Manley
			11	1,670	
	GEORGIA				
Op.	Atlantic Refg. Co.	Brunswick	6	2,000	Lewis
			6	2,000	
	ILLINOIS				
Op.	Indian Refg. Co.	Lawrenceville	8	8,000	Cross
Op.	Indian Refg. Co.	Lawrenceville	4	2,000	Dubbs
Op.	Lemont Refg. Co.	Lemont	2	1,500	Donnelly
Op.	Lincoln Oil Refg. Co.	Robinson	4	6,000	Holmes-Manley
Op.	Lubrite Refg. Co.	E. St. Louis	3	750	Fleming
SD.	Lubrite Refg. Co.	E. St. Louis	1	250	Fleming
Op.	Shell Pet. Corp.	Wood River	2	2,800	Cross
Rebldg.	Shell Pet. Corp.	Wood River	2	1,200	Cross
Op.	Shell Pet. Corp.	Wood River	16	14,000	Dubbs
Op.	Standard Oil Co.(Ind.)	Wood River	50	10,833	Burton
SD.	Standard Oil Co.(Ind.)	Wood River	10	2,167	Burton
Op.	Standard Oil Co.(Ind.)	Wood River	6	13,000	Holmes-Manley
Op.	The Texas Co.	Lockport	6	5,500	Holmes-Manley
Op.	White Star Refg. Co.	Hartford	3	2,600	Dubbs
			117	70,600	
	INDIANA				
Op.	Bartles-Maguire Oil Co.	E. Chicago	2	4,000	Jenkins
Op.	Shell Pet. Corp.	E. Chicago	6	8,000	Dubbs
Bldg.	Shell Pet. Corp.	E. Chicago	2	3,000	Dubbs
Op.	Sinclair Refg. Co.	E. Chicago	40	-40,000	Isom
Op.	Standard Oil Co.(Ind.)	Whiting	40	8,800	Burton
SD.	Standard Oil Co.(Ind.)	Whiting	146	24,800	Burton
Bldg.	Standard Oil Co.(Ind.)	Whiting	2	6,000	Cross
Op.	Standard Oil Co.(Ind.)	Whiting	22	59,000	Holmes-Manley
			260	153,600	
	KANSAS				
SD.	Arkansas City Refg.Co.	Arkansas City	2	1,500	Dubbs
Op.	Barnsdall Refineries, Inc.	Wichita	4	1,500	Dubbs
Op.	Derby Oil & Refg.Corp.	Wichita	4	2,200	Dubbs

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	KANSAS (Continued)				
Op.	Golden Rule Refg. Co.	Wichita	1	1,200	Jenkins
Bldg.	Hutchinson Oil Refg.Co.	Hutchinson	1	500	Trumble
Op.	Kanotex Refg. Co.	Arkansas City	3	3,000	Jenkins
SD.	Miller Pet. Co.	Humboldt	1	500	Cross
Op.	National Refg. Co.	Coffeyville	1	1,600	Setzler
Op.	The Peerless Oil & Refg. Co.	Chanute	2	1,500	Jenkins
Op.	Shell Pet. Corp.	Arkansas City	8	8,000	Dubbs
Bldg.	Shell Pet. Corp.	Arkansas City	2	2,000	Dubbs
Op.	Sinclair Refg. Co.	Argentine	10	10,000	Isom
Op.	Sinclair Refg. Co.	Coffeyville	10	10,000	Isom
Op.	Skelly Oil Co.	Eldorado	12	11,000	Jenkins
Op.	Skelly Oil Co.	Eldorado	2	2,600	Winkler-Koch
Op.	Standard Oil Co.(Kans.)	Neodesha	20	3,000	Burton
SD.	Standard Oil Co.(Kans.)	Neodesha	40	6,000	Burton
Op.	Standard Oil Co.(Kans.)	Neodesha	1	2,500	Holmes-Manley
Op.	Vickers Pet. Co.	Potwin	2	1,500	Dubbs
Op.	White Eagle Oil & Refg. Co.	Augusta	20	3,000	Burton
			146	73,100	
	KENTUCKY				
Op.	Aetna Oil Service,Inc.	Louisville	2	400	Fleming
Op.	Ashland Refg. Co.,Inc.	Leach	1	1,000	Dubbs
Bldg.	Latonia Refg. Corp.	Latonia	1	1,200	Tube and Tank
Op.	Louisville Refg.Co., Inc.	Louisville	2	1,600	Dubbs
Op.	Standard Oil Co. (Ky.)	Louisville	30	3,000	Burton
Op.	The Texas Co.	Pryse	1	2,000	Holmes-Manley
			37	9,200	
	LOUISIANA				
Op.	Corco Oil Refg. Corp.	Cedar Grove	2	3,400	Jenkins
Bldg.	Dixie Oil Co., Inc.	Superior	1	2,000	Cross
Op.	Louisiana Oil Refg.Corp.	Bossier City	4	4,000	Beacon-Coil
Op.	Louisiana Oil Refg.Corp.	Bossier City	5	6,000	Tube and Tank
Op.	New Orleans Refg. Co.	Good Hope	4	2,000	Dubbs
Bldg.	New Orleans Refg. Co.	Good Hope	4	6,000	Dubbs
Op.	Shreveport-Eldorado P. L. Co.	Shreveport	2	2,000	Dubbs

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	LOUISIANA (Continued)				
Op.	Shreveport-Eldorado P. L. Co.	Shreveport	1	1,000	Jenkins
Op.	Standard Oil Co.of La.	Baton Rouge	2	5,000	Cross
Op.	Standard Oil Co.of La.	Baton Rouge	16	38,000	Link
Bldg.	Standard Oil Co.of La.	Baton Rouge	4	15,000	Link
Op.	Standard Oil Co.of La.	Baton Rouge	8	12,000	Tube and Tank
			53	96,400	
	MARYLAND				
SD.	Interocean Oil Co. of Del.	Baltimore	2	1,000	Dubbs
SD.	Interocean Oil Co. of Del.	Baltimore	4	2,000	Leamon
Bldg.	Prudential Refg. Corp.	Baltimore	2	4,000	Cross
SD.	Prudential Refg. Corp.	Baltimore	4	2,400	Dubbs
Op.	Standard Oil Co. of N. J.	Baltimore	8	13,600	Tube and Tank
SD.	Standard Oil Co. of N. J.	Baltimore	4	6,800	Tube and Tank
			24	29,800	
	MASSACHUSETTS				
Op.	Beacon Oil Co., Inc.	Everett	14	25,000	Tube and Tank
Bldg.	Cities Service Refg.Co.	E. Braintree	2	4,500	Doherty
Op.	Cities Service Refg.Co.	E. Braintree	2	2,500	Holmes-Manley
			18	32,000	
	MICHIGAN				
Bldg.	White Star Refg. Co.	Trenton	2	3,600	Dubbs
			2	3,600	
	MISSOURI				
SD.	Standard Oil Co.(Ind.)	Sugar Creek	20	4,286	Burton
SD.	Standard Oil Co.(Ind.)	Sugar Creek	60	7,500	Burton
Op.	Standard Oil Co.(Ind.)	Sugar Creek	4	7,857	Holmes-Manley
Op.	Wilhoit Refg. Co.	Joplin	1	1,000	Jenkins
			85	20,643	
	MONTANA				
Op.	Arro Oil & Refg. Co.	W. Lewistown	1	500	Dubbs
			1	500	

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	NEW JERSEY				
Op.	Bertrin Pet. Co.	Maurer	1	3,000	Cross
Rebldg.	Eastern Oil Processing Co.	Petty Island	2	1,700	Doherty
Op.	Gulf Refg. Co.	Bayonne	1	1,000	de Florez
Op.	Standard Oil Co. of N. J.	Bayonne, etc.	40	10,000	Burton
SD.	Standard Oil Co. of N. J.	Bayonne, etc.	40	10,000	Burton
Op.	Standard Oil Co. of N. J.	Bayonne, etc.	21	36,500	Tube and Tank
Bldg.	Standard Oil Co. of N. J.	Bayonne, etc.	6	18,000	Tube and Tank
SD.	Standard Oil Co. of N. J.	Bayonne, etc.	6	9,000	Tube and Tank
Op.	Tide Water Oil Co.	Bayonne	4	6,500	Tube and Tank
Op.	Vacuum Oil Co.	Paulsboro	3	2,400	Cross
Op.	Vacuum Oil Co.	Paulsboro	3	1,800	Tube and Tank
SD.	Warner-Quinlan Co.	Warners	2	1,200	Dubbs
			129	101,100	
	NEW YORK				
Op.	Standard Oil Co. of N. Y.	Brooklyn and L. I. City	6	8,000	Cross
Op.	Standard Oil Co. of N. Y.	Buffalo	2	2,000	Cross
Op.	Vacuum Oil Co.	Olean	2	1,200	Cross
Op.	Vacuum Oil Co.	Olean	1	600	Tube and Tank
			11	11,800	
	OHIO				
Op.	Paragon Refg. Co.	Toledo	2	4,000	Dubbs
Op.	Pure Oil Co.	Heath	2	4,000	Cross
Op.	Pure Oil Co.	Heath	1	500	Gyro
Bldg.	Pure Oil Co.	Heath	1	500	Gyro
SD.	Solar Refg. Co.	Lima	15	5,025	Burton
Op.	Solar Refg. Co.	Lima	2	2,000	Cross
Op.	Solar Refg. Co.	Lima	30	10,050	Burton
Op.	Standard Oil Co. of Ohio	Cleveland	2	3,200	Tube and Tank
Bldg.	Standard Oil Co. of Ohio	Cleveland	2	3,200	Tube and Tank
Op.	Standard Oil Co. of Ohio	Toledo	2	3,200	Tube and Tank
Op.	Stellar Refg. Co.	Marne	1	500	Leamon
			60	36,175	

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	OKLAHOMA				
Op.	Barnsdall Refineries, Inc.	Barnsdall	3	4,000	Cross
Op.	Barnsdall Refineries, Inc.	Okmulgee	3	3,500	Cross
Op.	Beckett Co.	Beckett	1	1,500	Jenkins
Op.	Bell Oil & Gas Co.	Grandfield	1	600	Dubbs
Op.	Bilmont Refg. Co.	Garber	1	1,000	Jenkins
SD.	Bolene Refg. Co.	Enid	2	2,000	Jenkins
Op.	Champlin Refg. Co.	Enid	2	2,000	Cross
Op.	Continental Oil Co.	Sapulpa	6	3,600	Cross
SD.	Empire Oil & Refg. Co.	Cushing	2	1,500	Doherty
Op.	Empire Oil & Refg. Co.	Okmulgee	1	1,000	Doherty
Op.	Empire Oil & Refg. Co.	Ponca City	3	2,500	Doherty
Op.	Empire Oil & Refg. Co.	Ponca City	2	2,000	Dubbs
Op.	Globe Oil & Refg. Co.	Blackwell	1	1,000	Cross
Op.	Imperial Refg. Co.	Ardmore	4	2,000	Dubbs
Op.	Independent Oil & Gas Co.	Okmulgee	1	1,250	Jenkins
Op.	Johnson Oil Refg. Co.	Cleveland	4	2,700	Dubbs
Op.	Marland Refg. Co.	Ponca City	2	7,000	Cross
Op.	Marland Refg. Co.	Ponca City	6	9,000	Dubbs
Op.	Marland Refg. Co.	Ponca City	12	3,600	Fleming
Op.	Mid-Continent Pet.Corp.	W. Tulsa	70	12,000	Coast
SD.	Mid-Continent Pet.Corp.	W. Tulsa	10	2,000	Coast
Op.	Mid-Continent Pet.Corp.	W. Tulsa	4	4,500	Koontz
Op.	Pierce Petroleum Corp.	Sand Springs	4	4,000	Cross
Op.	Producers & Refiners Corp.	W. Tulsa	6	3,000	Dubbs
Op.	Pure Oil Co.	Ardmore	4	2,000	Dubbs
Op.	Pure Oil Co.	Muskogee	2	4,000	Cross
Op.	Shaffer Oil & Refg.Co.	Cushing	5	3,800	Dubbs
Op.	The Texas Co.	W. Tulsa	12	16,800	Holmes-Manley
SD.	Texas Pacific Coal & Oil Co.	Wynnewood	2	1,000	Dubbs
Op.	Tidal Refg. Co.	Drumright	2	3,000	Tube and Tank
Op.	Transcontinental Oil Co.	Boynton	6	900	Slagter
Op.	Transcontinental Oil Co.	Bristow	9	1,350	Slagter
Bldg.	White Oak Refg. Co.	Allen	1	1,400	Jenkins
Bldg.	H.F.Wilcox Oil & Gas Co.	Bristow	1	2,000	Dubbs
			195	113,500	

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	PENNSYLVANIA				
Op.	American Oil Works Co.	Titusville	1	600	Fleming
Op.	Atlantic Refg. Co.	Franklin	6	6,000	Cross
Op.	Atlantic Refg. Co.	Philadelphia	10	20,000	Cross
SD.	Atlantic Refg. Co.	Philadelphia	32	12,000	Lewis
Op.	Atlantic Refg. Co.	Pittsburgh	2	4,000	Cross
Op.	Butler County Oil Refg. Co.	Bruin	4	2,000	Bruin
Op.	Emlenton Refg. Co.	Emlenton	2	200	Bruin
Op.	Freedom Oil Works Co.	Coraopolis	1	400	Dubbs
SD.	Hartol Products Corp.	Franklin	1	600	Richie
Bldg.	General Benzol Corp.	Warren	4	1,000	Beggs
Bldg.	General Benzol Corp.	Warren	4	1,000	Armont
Op.	Independent Refg. Co.	Oil City	4	400	Bruin
SD.	Island Pet. Co.	Neville Island	2	300	Bruin
Op.	Kendall Refg. Co.	Bradford	1	650	Dubbs
SD.	Kendall Refg. Co.	Bradford	1	500	Dubbs
Op.	Mutual Refg. Co.	Warren	2	550	Bruin
Op.	Penn. Oil Products Refg. Co.	Eldred	2	1,000	Bruin
Op.	The Pennzoil Co.	Oil City	2	1,800	Dubbs
Op.	Pure Oil Co.	Marcus Hook	4	8,000	Cross
Bldg.	Pure Oil Co.	Marcus Hook	3	3,000	Gyro
Op.	Sinclair Refg. Co.	Marcus Hook	20	20,000	Isom
Op.	Sun Oil Co.	Marcus Hook	7	8,000	Cross
Op.	Sun Oil Co.	Marcus Hook	1	8,000	Own
Bldg.	Sun Oil Co.	Marcus Hook	2	8,000	Own
SD.	Swan-Finch Refg. Co.	Warren	2	517	Bruin
Op.	Titusville Oil Works	Titusville	1	200	Fleming
Op.	Valvoline Oil Refinery	E. Butler	4	400	Bruin
Op.	Waverly Oil Works Co.	Coraopolis	1	3,000	Cross
Op.	Waverly Oil Works Co.	Pittsburgh	1	800	Cross
			127	112,917	
	RHODE ISLAND				
Op.	Standard Oil Co.of N.Y.	E. Providence	3	6,000	Cross
			3	6,000	
	SOUTH CAROLINA				
Op.	Standard Oil Co.of N.J.	Charleston	8	10,000	Tube and Tank
			8	10,000	

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	TEXAS				
Op.	American Refg. Properties	Wichita Falls	2	2,000	Cross
Op.	American Refg. Properties	Wichita Falls	4	3,000	Dubbs
Op.	Continental Oil Co.	Wichita Falls	4	2,400	Dubbs
Bldg.	J. S. Cosden & Co.	Big Spring	4	6,000	Jenkins
Op.	Crown Central Pet.Corp.	Houston	3	5,000	Holmes-Manley
SD.	Empire Oil & Refg.Co.	Gainesville	1	1,000	Doherty
Op.	Empire Oil & Refg. Co.	Gainesville	2	2,000	Dubbs
Op.	Grayburg Oil Co.	San Antonio	2	1,500	Dubbs
Op.	Gulf Refg. Co.	Fort Worth	6	4,200	Buerger
SD.	Gulf Refg. Co.	Port Arthur	11	7,700	Buerger
Op.	Gulf Refg. Co.	Port Arthur	70	39,700	Buerger
Bldg.	Gulf Refg. Co.	Sweetwater	2	4,000	Buerger
Op.	Humble Oil & Refg. Co.	Baytown	2	5,000	Cross
Op.	Humble Oil & Refg. Co.	Baytown	20	34,000	Tube and Tank
Bldg.	Humble Oil & Refg. Co.	Ingleside	2	6,000	Tube and Tank
Bldg.	Humble Oil & Refg. Co.	McCamey	2	6,000	Tube and Tank
Op.	Magnolia Pet. Co.	Beaumont	80	10,000	Burton
Op.	Magnolia Pet. Co.	Beaumont	16	37,500	Cross
Op.	Magnolia Pet. Co.	Fort Worth	1	2,000	Cross
Op.	Motor Fuel Products Co. Inc.	Laredo	1	1,500	Jenkins
SD.	Oriental Oil Co.	W. Dallas	3	300	Floyd
Op.	Panhandle Refg. Co.	Wichita Falls	2	1,400	Dubbs
Bldg.	Pecos Refg. Co.	Pecos	2	3,000	Jenkins
Op.	Petroleum Conversion Corp.	Texas City	1	300	Knox
Op.	Pure Oil Co.	Nederland	26	13,000	Cross
Op.	Pure Oil Co.	Nederland	3	3,000	Gyro
Bldg.	Richardson Refg. Co.	Big Spring	2	2,800	Jenkins
Op.	Simms Oil Co.	W. Dallas	4	3,600	Cross
Op.	Sinclair Refg. Co.	Sinco	20	20,000	Isom
Op.	Star Refg. & Producing Co.	Fort Worth	1	200	Fleming
SD.	J.J.& M. Taxman Refg.Co.	Wichita Falls	1	650	Cross
Op.	Terminal Oil & Refg.Co.	Texas City	1	1,800	Jenkins
Bldg.	The Texas Co.	El Paso	1	1,650	Holmes-Manley
Op.	The Texas Co.	Gates	2	3,000	Holmes-Manley
SD.	The Texas Co.	Houston	24	6,600	Burton
SD.	The Texas Co.	Houston	1	1,714	Tube and Tank

Status	Company	Location	Number of units	Total daily charging capacity, barrels	Type of process
	TEXAS (Continued)				
Op.	The Texas Co.	Port Arthur	74	79,200	Holmes-Manley
Bldg.	The Texas Co.	San Antonio	2	3,300	Holmes-Manley
Op.	Texas Pacific Coal & Oil Co.	Fort Worth	1	600	Cross
Op.	Transcontinental Oil Co.	Fort Worth	9	1,350	Slagter
SD.	White Eagle Oil & Refg. Co.	Fort Worth	1	1,500	Holmes-Manley
			416	329,464	
	UTAH				
Op.	Utah Oil Refg. Co.	N. Salt Lake City	1	2,500	Burton
SD.	Utah Oil Refg. Co.	N. Salt Lake City	1	2,000	Burton
Op.	Utah Oil Refg. Co.	N. Salt Lake City	2	5,000	Holmes-Manley
			4	9,500	
	WEST VIRGINIA				
Op.	Ohio Valley Refg. Co.	St. Marys	3	300	Bruin
Op.	Pure Oil Co.	Cabin Creek Jct.	3	1,200	Gyro
SD.	Standard Oil Co.of N.J.	Parkersburg	6	1,500	Burton
Op.	Standard Oil Co.of N.J.	Parkersburg	2	3,000	Tube and Tank
Op.	Tri-State Refg. Co.	Kenova	1	2,000	Jenkins
			15	8,000	
	WYOMING				
Op.	Continental Oil Co.	Glenrock	10	1,690	Burton
SD.	Continental Oil Co.	Glenrock	5	1,150	Burton
SD.	Egaso Holding Corp.	Osage	1	1,000	Cross
SD.	Midwest Refg. Co.	Casper	40	6,200	Burton
Op.	Midwest Refg. Co.	Greybull	30	3,510	Burton
SD.	Midwest Refg. Co.	Greybull	10	1,170	Burton
Op.	Midwest Refg. Co.	Laramie	10	1,550	Burton
SD.	Midwest Refg. Co.	Laramie	10	1,550	Burton
Op.	Producers & Refiners Corp.	Parco	4	2,800	Dubbs
SD.	Producers & Refiners Corp.	Parco	2	1,400	Dubbs
Op.	Standard Oil Co.(Ind.)	Casper	187	31,855	Burton

Status	Company	Location	Number of units	Total daily charging capacity barrels	Type of process
	WYOMING (Continued)				
SD.	Standard Oil Co.(Ind.)	Casper	60	7,680	Burton
Op.	The Texas Co.	Casper	6	9,400	Holmes-Manley
Bldg.	The Texas Co.	Cody	1	1,600	Holmes-Manley
Op.	White Eagle Oil & Refg. Co.	Casper	1	1,800	Holmes-Manley
Op.	White Eagle Oil & Refg. Co.	Casper	1	2,000	Jenkins
			378	76,355	

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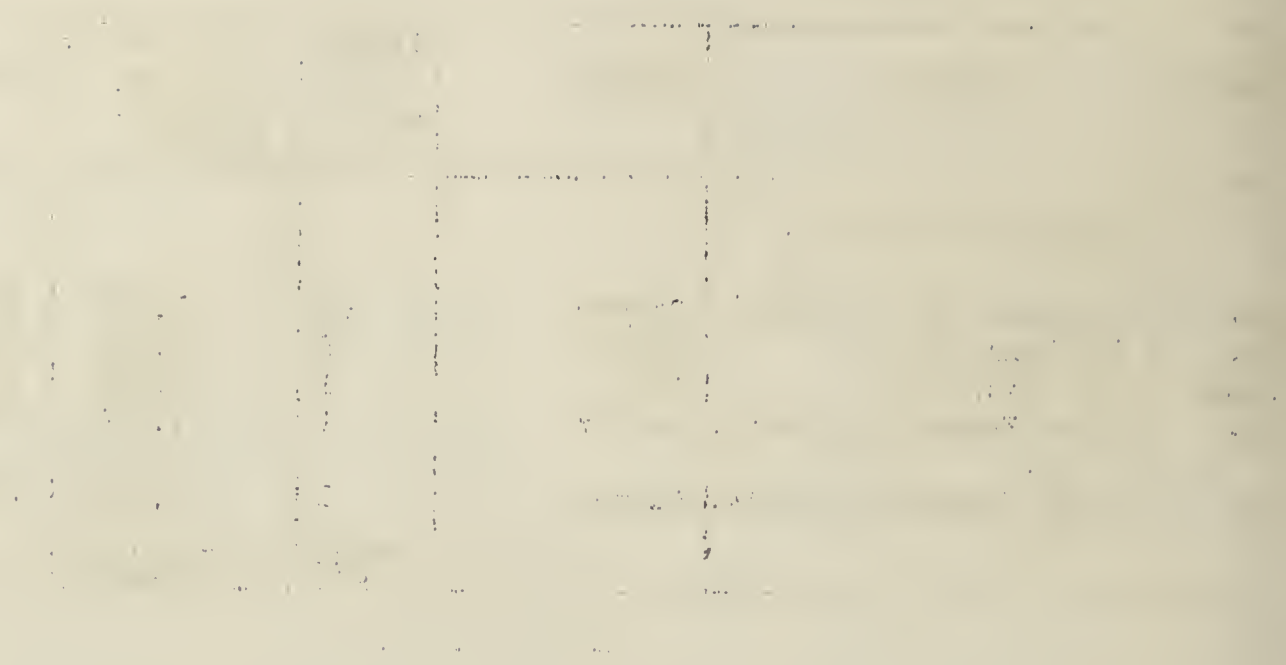

Circular 6130 May, 1929

INFORMATION CIRCULAR

Department of Commerce -- Bureau of Mines

THE UNUSUALLY GOOD SAFETY RECORD OF A COAL MINE AND OF A COAL-MINE FOREMAN

BY

E. H. Denny

Circular 6130,
May, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

THE UNUSUALLY GOOD SAFETY RECORD OF A COAL MINE AND OF A COAL MINE FOREMAN [1]

By E. H. Denny[2]

A most interesting safety record has been made by the Colorado Fuel and Iron Co.'s. Robinson No. 1 mine located at Walsen, Colo. Mr. J. L. McBrayer has been superintendent of the Robinson No. 1 and 2 mines for the past 2½ years, and Mr. David Muir has been foreman of the Robinson No. 1 mine since June, 1913.

The last fatal accident at Robinson No. 1 mine occurred on July 27, 1915, and the tonnage produced at this mine since July 28, 1915, has been as follows:

	Tons
July 28, 1915, to Dec. 31, 1915	142,378
1916	210,795
1917	215,773
1918	219,187
1919	147,695
1920	204,150
1921	142,294
1922	166,283
1923	151,935
1924	137,321
1925	169,548
1926	203,265
1927	180,389
Jan. 1, 1928, to Feb. 21, 1929	204,042
Total	2,475,555

In 1928 the Joseph A. Holmes Safety Association awarded a Joseph A. Holmes Certificate on the following basis:

> "From July 27, 1915 to March 1, 1928, the Robinson No. 1 mine operated without a single fatal accident, and during that time a total of 2,301,804 tons of coal were mined."

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1 The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be used: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

2 District engineer, safety division, U. S. Bureau of Mines Field Office, Denver, Colo.

At the March, 1929, meeting of the Joseph A. Holmes Safety Association in Washington, D. C. a special certificate was awarded to Mr. David Muir, mine foreman of the Robinson No. 1 mine, because of the most excellent safety record of the mine under his direct supervision for so many years. The basis of this 1929 award of the certificate to a person was the fact that investigation indicated that the good record of the mine was in a large measure due to Mr. Muir's own efforts.

The safety rules in force at the Robinson No. 1 mine do not appear to differ materially from those at any of the other mines of the company, nor are the safety measures or operating conditions at this mine particularly different from other mines of the company. The company has a limited number of definite safety rules which during the past two years they have made redoubled efforts to enforce in all mines. Systematic timbering practice is required in all mines of the company; thus in the two Robinson mines props are set 7 feet or less from the face and 6 feet apart; they are set closer if necessary. In machine work the machineman sets a temporary prop first, then removes the standard prop, and after cutting resets the original prop. In pick-mining, spragging of the coal is required. During his examination the fire boss marks, with a circle and cross, points where timbers are needed. If when the foreman or assistant foreman visits the place the prop is not in the point designated by the fire boss or at least in the process of being set, the man responsible goes home. The fire boss notifies the men of any timbers shot out from blasting and of any dangers noted in the working places. Machinemen are certified men and carry safety lamps.

At the Robinson mines the men are required to travel on the manways, and unauthorized persons traveling on haulage slopes are severely disciplined. Mule haulage is used in the Robinson No. 1 mine, except for rope haulage on the slope. Drags are used on trips wherever there is a hoist, and a rope rider is discharged if he does not use the drag. The use of a tie and crosstie in car-blocking is standard practice. Violation of this practice is followed by immediate discipline.

Man trips are secured by additional fastening to the main hoisting rope which passes along the outside of the cars and fastens to the rear car. The man trip is used to take men out of the mine, and the men travel into the mine on the manways.

The Robinson No. 1 and 2 mines are now worked entirely by the room-and-pillar system. Rooms are driven 25 feet wide and 250 feet long, with 35 feet of pillar between rooms. The coal is about 5 feet in height. All advance work is cut by machine, as are most of the pillars.

At some places in the mines a good roof is formed by sandstone but in other places where a shale occurs the roof is only fairly good. In the No. 1 mine there is sometimes slate which has to be taken down after the cut. In No. 2 mine there is sometimes 4 feet of draw slate which in many places is held up.

Cutting machines in these mines use 440-volt alternating cuttent; 500-volt direct current is used on haulage. Endeavor is made to guard wires at possible contact points. Bare points on the power lines for connecting the "nips" are now protected by wooden guards.

Classification of the 216 employees of the Robinson No. 1 mine according to nationality is as follows:

	Number	Per cent
Mexican	57	26.4
American white.......	55	25.5
Austrian	34	15.7
Italian	26	12.0
American negroes	16	7.4
Polish	6	2.8
German	4	1.9
Miscellaneous	18	8.3
	216	100.0

About the same condition obtains as to nationality of employees at the other mines of the company, except that at a few there is a somewhat higher percentage of Mexican labor.

Under the company joint-representation plan a grievance committee of foremen and employees' representatives meets monthly. Two employees' representatives are elected from each mine by the employees each year. At the Robinson mines it is stated that there have been practically no grievances and that all the monthly meetings have, therefore, been turned into safety meetings. At the other mines of the company the same meetings are also devoted largely to safety.

When a fatal accident occurs in the Colorado Fuel and Iron Co. mines, it is investigated by the safety and accident committee comprising three representatives of the employees and three members of the management. When a fatal accident occurs, the details are sketched by an engineer from the division engineer's office, and blue prints of such sketches are made available for the general information of officials and employees. A hearing or "court" is held by company officials on serious and fatal accidents, this body consists usually of the general superintendent of the company, the company mine inspector, and the mine superintendent. The court is held with the idea of fixing responsibility for the accident and of making a thorough investigation of the accident.

Several of the mines have general safety meetings of employees at frequent intervals, and almost 100 per cent attendance of employees is obtained.

Every man with the Colorado Fuel and Iron Co. carries a pocket first-aid packet purchased at cost from the company store. He is required to have such packet with him all of the time. The company now requires that all officials and other employees shall receive first-aid training once a year; during a large part of the time classes in first aid are being carried on at several of their mines. The Colorado Fuel and Iron Co. rescue car and the Bureau of Mines safety car spend one or more weeks at each of the company's operations yearly, giving instruction in first aid and training selected men of the company in the use of mine-rescue apparatus. In addition, a considerable number of officials and selected men of the company have been given the bureau's advanced course in rescue and recovery operations for mine explosions and fires. In its various fields the company maintains several mine-rescue stations equipped with mine-rescue apparatus and accessories and with the all-service or type N gas mask, in addition to maintaining a mine-rescue railroad car and a fire fighting car. Trained rescue crews are also available at these mines.

Some employees of the company, particularly miners, are required to use goggles during certain operations; for example, a miner is required to use goggles during all pick work. Search of men for goggles and first-aid packets is occasionally made the same as for matches. If a miner sustains an eye injury he is disciplined and this rule has made it possible to secure the general use of goggles.

Permissible electric lamps are used in the company's coal mines.

Recently the Colorado Fuel and Iron Co. adopted the plan of giving safety talks in the working places of its coal mines. Various designated officials of the company prepare a short talk in mimeograph form which is distributed to all foremen and assistants in order that they may familiarize themselves with it. The foremen and assistants take about 5 minutes in each working place at least every two weeks to give the worker the substance of such prepared safety talks. If the worker does not speak English, or the boss does not speak the particular foreign language of the worker, an interpreter is used.

As stated previously, these practices are very largely standard in the coal operations of the company. Discipline takes the form ordinarily for a first offense of two or three days' lay-off, and for a second or sometimes a third offense, of discharge. During the past year approximately 31 per cent reduction in the frequency rate of accidents was secured by the company.

The exceptional record of the Robinson No. 1 mine is credited by the mine superintendent and other officials to the work of David Muir, mine foreman. Mr. Muir has had a total of about 45 years' mining experience and has been employed by the Colorado Fuel and Iron Co. for more than 38 years. It is said that work under Mr. Muir's direction during this period has been characterized by a remarkably low accident rate, and that he has been particularly happy in having at all

times the confidence and good will of his men. He has made it plain that company rules as well as his own and other officials' orders must be rigidly obeyed. By the use of plainly worded instructions spoken in a calm tone of voice he has been able to make the men understand that orders must be put into immediate effect.

The Robinson No. 2 mine does not differ essentially from the No. 1 mine, as the two mines connect and work in the same seam, under the same roof, and with essentially the same methods and class of labor; general conditions are considered to be no worse in one than in the other of the mines. Until recently, nevertheless, the No. 2 mine had a relatively poor safety record, whereas the No. 1 mine under Mr. Muir had a very good record. The good safety record of the No. 1 mine is considered to be due to company rules, careful supervision, proper discipline of careless or indifferent workmen, and personal contact of the mine foreman with the men, as carried out by Mr. Muir. Evidently it has been made manifest to all company officials and employees at the Robinson No. 1 mine that safety is at least of equal importance with production.

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Circular No. 6131
May, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

IX. MINING LAWS OF COLOMBIA[1]

By A. D. Garman[2]

PREFATORY NOTE

This paper presents one of a series of digests of foreign mining legislation and court decisions which is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the laws of Colombia has been prepared from the best available information in Washington, but is released subject to correction and amplification, if necessary, by the proper American diplomatic and consular officers to whom it is being referred through the courtesy of the Department of State.

SYNOPSIS OF LAW

Minerals in Colombia belong to the owner of the soil unless otherwise provided in the law. The following lands and deposits, however, are described as fiscal property of the State:

Class 1. The public lands, mines, and salines which belong to the individual States (now Departments) prior to the adoption of the present constitution (1886).

Class 2. All deposits of gold, silver, platinum and precious stones.

Class 3. All copper deposits.

Class 4. All other mineral deposits in public lands, such as coal, iron, sulphur, petroleum, asphalt, etc.

Class 5. All deposits of guano and other fertilizers discovered on lands which either are or have been public.

By grants, or adjudications, under existing law, ownership of the surface and subsoil is separate. But title to the mine always carries with it the right

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2 Principal translator, U. S. Bureau of Mines, Washington, D. C.

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to use the land necessary to work the mine (Art. 3). The owner of the mine is responsible, however, for damages to the surface-owner (Art. 191 ff.).

Rights of Aliens

Foreigners in Colombia enjoy the same rights as are granted to Colombians by the laws of the nation to which the foreigner belongs -- subject to stipulations in public treaties to the contrary (National Constitution, Art. 11). Law 72 of 1910, however, contains a local and temporary discrimination against aliens in the following terms: "Upon the ratification of the present law, and while the codes and laws on mines and public lands are undergoing revision and reform...all adjudication of nationally owned property to foreign individuals, entities, or companies in the region of Chocó and Darien shall be suspended."

Foreigners need not incorporate under the laws of Colombia, but foreign companies must be registered there. (Legislative Decrees Nos. 2 and 37 of 1906.) Companies or corporations recognized as juristic persons have no other rights than those pertaining to Colombian individuals. (Constitution, Art. 14.)

Prospecting

The only general requirement for securing possession and ownership of mines is the "legal capacity for acquiring dominion over things" pursuant to the usual laws (Art. 2).

By later legislation, however, emeralds, petroleum, hydrocarbons in general, and salt are excepted from denouncement, and are either government monopolies or are worked by contract with the State.

There are also certain local exceptions with respect to other minerals:

1. Gold and silver mines which have been exploited on behalf of the nation in Marmota, Supia, and Santa Anna. (Law 38 of 1887, Art. 6.)

2. Gold placer mines within the limits of patented vein mines which have paid the tax established by law. (Law 272 of 1875, Art. 52.)

Certain areas are excepted:

(a) The beds of navigable streams. (Law 72 of 1910, Art. 5.)

(b) The bed and banks of the River Cauca, up to high-flood limit. (Law 38 of 1887.)

(c) Lands belonging to educational or charitable institutions, except with the authorization of the respective owners. (Decree No. 1112 of 1905, Art. 5.)

(d) Lands on which mines reserved by the nation are situated. (Legislative Decree No. 48 of 1905, Art. 1.)

The following exceptions apply more particularly to prospecting on private property:

(a) Within the area of a settlement and within 100 meters from its most outlying houses, unless the working of the mine be in a direction away from such settlement and without probable damage thereto, proximate or remote. In such cases permission may be granted by the police.

(b) Within the courts, gardens, orchards, or lawns of rural dwellings, except by the owner.

(c) In other inclosed and permanently cultivated lands, except by giving previous notice to the owner and furnishing security if required.

The owner's permission is also required in connection with:

(a) Alluvial deposits on privately owned land which is cultivated or used for cattle-raising. (Law 38 of 1887, Art. 3.)

(b) Lode deposits on privately owned land in certain Departments where by prior laws the proprietor of the soil owned also the subsoil, when such land is cultivated or used for cattle-raising.

Prospecting itself gives no preferential right. This belongs to the first discoverer, who is obliged to give notice of discovery to the municipal chief of the district (or one of the districts) where the mine is situated and to make a formal denouncement of the mine within 90 days after the discovery is deemed to have been made.

Classification and Area of Mines

Mines or mineral deposits are divided into three classes according to their formation: (1) Lode deposits, such as those of precious stones, silver, and gold; (2) sedimentary deposits, such as are ordinarily those of iron and copper; and (3) alluvial deposits or placers.

The discoverer of a lode is entitled to an area not exceeding 1,800 by 240 meters (i.e., three pertenencias, each 600 by 240 meters, measured upon the surface and not on a horizontal plane.

For placers the area is either a square with a base of 3 kilometers or a rectangle 2 by 5 kilometers.

For sedimentary deposits, the area is a square with a base of 2 kilometers.

Denouncers of lode mines situated in lands belonging to the State may secure up to 500 hectares in land continuous with the claims to which they are entitled by law. (Law 75 of 1887, Art. 1.)

Concessions

In order to work the deposits of classes 4 and 5, contracts must be entered into with the Government. These contracts do not require ratification by Congress, provided that:

1. The duration does not exceed 30 years (petroleum, 55; coal and fertilizers, 50).

2. The State be given free title at the expiration of the contract to all the transportation facilities and articles used in the operation of the mine.

3. The State's share of the gross production be at least 15 per cent.

In the granting of such concessions, no limitation is imposed upon area, and no percentage of ownership by nationals is stipulated. Monopolistic concessions, however, are not expressly provided for, and Legislative Act No. 3 of 1910, Art. 4 (incorporated in the Constitution under Art. 31) declares that no monopoly may be established except as a revenue expedient and by virtue of law.

Fees and Taxes

The proceedings for securing possession of mines are taken at the expense of the denouncers, who are obliged to furnish the public official and two experts who participate therein with the food and vehicles required. They also pay, by way of fees, to the official, 80 centavos for each myriameter (= 6.2137 miles) of distance he is obliged to travel in order to reach the spot, and 1 peso for the act of giving possession. To each expert they pay 40 centavos for each hour's work and 80 centavos for each myriameter of distance. (Law 292 of 1875, Art. 14.)

Other fees and taxes are as follows:

For the denouncement of a gold or silver mine, 50 centavos.

For the title of concession of such a mine, 4 pesos gold.

For every vein mine of gold or silver, whether worked or not, an annual tax of 1 peso gold for each claim (pertenencia).

Any such mines having an area either greater or less than one claim, pay 1 peso gold annually per claim or fraction thereof.

On each gold placer mine of the regulation size and shape -- that is, a square with a base of 5 kilometers (Art. 28) -- 1 peso gold annually. (Law 59 of 1909, Art. 2.)

On each mine of precious stones with an area not exceeding 1 square kilometer, 5 pesos annually, and on any excess area in the same proportion.

On copper mines, one-half of the duties paid on mines of precious stones. (Law 21 of 1909, Art. 8.)

Law 292 of 1875, Art. 24, provides that the owner of a mine can abandon any determinate part of it and thereafter pay the tax only on the part which he retains.

Perpetual ownership of a mine, and freedom thereafter from taxation, can be obtained by paying in a lump sum twice the amount which would have to be paid in taxes in 20 years. (Law 59 of 1909, Art. 3.)

Cancellations

Patents or titles issued under the mining code are void in the following cases:

1. When the signature of the Governor of the Department and of the Secretary of Hacienda is lacking.

2. When the patents or titles have not been duly registered.

3. When a greater area is granted than the law allows.

4. When, in denouncing a mine, a name has been used which is different from that by which the region was known where the mine is situated.

5. When the name of the last possessor of the mine is omitted, although the denouncer knows it, and also when another is substituted for said name.

6. When any owner of an adjoining mine is not cited to appear at the time possession is given.

7. When the documents inserted in the title are not in accord with the originals from which they were copied.

All these nullities may be remedied, however, unless bad faith is proved on the part of the title-holder.

After discovery has been made, the right to the mine in question can be lost only in the following cases:

1. When the discoverer does not appear for the purpose of denouncing it within 90 days after the discovery.

2. When the discoverer has undertaken to deliver to the official commissioned for giving possession of the mine the instructions issued by the executive power, and fails to do so within the prescribed time limit(20 days plus ordinary traveling time).

3. When the discoverer fails to apply for the possession of the mine within 60 days from the expiration of the period (three weeks) for posting notice of the possession to be given.

4. When the discoverer fails to solicit the issuance of the title or patent within 60 days after receiving possession.

5. When the discoverer fails to pay the tax punctually.

6. When, having been cited to appear when possession of a mine is given, the discoverer makes no opposition and part or all of his own mine is comprised in the possession given (Art. 118).

These time limits are subject to extension, however, for cause shown.

Water Rights

The discoverer of the first mine in any region has a preferential right to the water required for an ordinary establishment, and may avail himself of this right at any time. Other discoverers have a like right, in strict order of seniority. The soil-owner, however, can not be deprived of the water required for his family, his animals, for irrigation, or for any kind of machinery set up or begun to be set up. (Arts. 205-208.)

Suspensions

Mining operations may be suspended or forbidden under the following circumstances:

If bond is not furnished, on application of the surface-owner and to the satisfaction of the municipal chief to whom such application is made, for the payment of damage resulting from the working of a mine (Art. 197).

In those mines whose working is prejudicial to public works, settlements, the waters used therein, and private dwellings, unless the owner or operator of such mines provides other suitable water and repairs at his expense the damage caused, so that such works may continue to operate without interruption.

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Oppositions

Oppositions may be made at any time after the denouncement of a mine is received up to the end of the period for posting; and by the owners of adjoining mines or by any one claiming superior right by reason of prior discovery, at the time possession of the new mine is given. In the former case, they may be made either before the official who has been commissioned for giving possession, the executive authority or the judge who is to take cognizance of the cause. In the latter case, the opposition must be formulated before the same judge within nine days plus ordinary traveling time reckoned from the date on which the possession was going to be given (Ch. VI). In all cases of opposition giving rise to a suit at law, the 60-day period in which to solicit possession is counted from the day when the official commissioned for this purpose received the court records of the case (Art. 57).

Miscellaneous

Export duties have been abolished. (Law 19 of 1909, Art. 1.) There are import duties of 1 per cent per kilogram on mining material. (Law 59 of 1909, Art. 1.)

Labor is scarce; foreign labor is excluded, though some of the conservative papers are advocating its admission.

Exchange

At par the Columbian peso is worth $0.9733 in United States currency, and recently it has been at a slight premium; the average exchange value for 1928, as reported by the Federal Reserve Board, was $0.9769.

CIRCULAR 6132 MAY, 1929

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

OCHER AND OCHERY EARTHS

BY

R. M. SANTMYERS

Circular No. 6132
May, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

OCHER AND OCHERY EARTHS[1]

By R. M. Santmyers[2]

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1 - The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be made: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

2 - Mineral specialist, rare metals and nonmetals division, U. S. Bureau of Mines.

GENERAL DESCRIPTION

Ocher or ochery earth is a natural mineral pigment composed largely of clay permeated with hydrated iron (ferric oxide).[3] Its color ranges from yellow through orange to reddish brown, depending largely on the amount of iron present. Ocher grades into sienna, which differs from ocher chemically in that it generally contains more ferric iron than it does clay, and physically in that when finely ground it is more of a stain than a pigment. Good grades of ocher should contain 20 or more per cent iron oxide, but the iron content and hence of the material marketed varies widely. Ocher has a specific gravity of about 3.5.

COLOR VARIATIONS

Cream ocher contains as low as 5 per cent iron hydroxide. It is used to some extent as a primer on wood, but has little value as a pigment.

Gray ocher is silica, clay, and carbonaceous matter. Sometimes it is colored slightly green by a trace of ferrous hydroxide. It is used as a filler for cheap paint.

White ocher is nothing but ordinary clay, and has no value in paint, although it is occasionally used as an adulterant.

Golden ocher is ocher which has been toned up with some chrome yellow. Various shades of it are on the market. Perfect orange-colored shades contain as much as 12 to 15 per cent chrome yellow. The base may be either French or domestic ocher.

Green ocher is similar in composition to gray ocher except that it contains a larger proportion of ferrous hydroxide. It is found principally in Bohemia (Czechoslovakia), and goes by the name of "terre verte." It has no hiding power when used alone in paints, but as it has a high absorbent capacity for certain aniline colors, it is largely used as a base for cheap lakes. Verona green, Veronese earth, green earth, etc., are similar products.

Red ocher is obtained by calcining raw ocher at a low heat so as to drive off a part of the combined water. The shade depends upon the time of heating and the iron content - the longer the calcination the more purple the product. Burnt ochers are sold as Indian red, Venetian red, light red, etc. As a rule, however, these ochers are much richer in iron than the ordinary ocher, which almost never contains more than 30 per cent iron oxide.

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3 - United States Tariff Commission, Tariff Information Survey, Par. 55, Act of 1913, A-15.

USES AND SPECIFICATIONS

There are two principal uses for ocher - in paints and as a filler for linoleum and oil cloth. Ocher is also used to a limited extent as a pigment in coloring cement stuccos and mortars, for producing desired colors in earthenware when mixed with manganese oxide, and in very limited quantities for other minor uses. It does not seem to follow that the best ocher for color pigments in paints is necessarily the most desirable as a filler in linoleum.

Domestic ochers of the best quality make excellent pigments, work well with all vehicles and with other pigments, and are permanent in color. When mixed with white, fine cream or buff tints are obtained. Chrome yellow is sometimes added to ocher of inferior color to impart a brighter tint than the natural color, but the chrome yellow fades after a time and leaves only the natural ocher effect.

Color would seem to be less important in ocher for use in linoleum and oil cloth, and this perhaps accounts for the fact that a large quantity of domestic ocher has been exported for use as a filler, giving as good service as the better colored but more expensive French ocher.

Georgia ocher has been used chiefly as a filler in the manufacture of linoleum and oil cloth, and much of it has been exported to England and Scotland for that purpose. Some of it is used for the same purpose in this country. It is also employed in the manufacture of paints, and in a variety of minor ways. This ocher when calcined yields a red pigment which is becoming of importance, especially as a mortar color, and is finding its way into the markets in growing quantities.

Pennsylvania ocher has been used principally in the manufacture of paints.

The United States Army states in its paint specifications that yellow ocher must be equal in color and quality to the best French ocher, must be free from chromate of lead or any foreign coloring matter, and must contain at least 20 per cent oxide of iron and not more than 5 per cent lime.

Other specifications prepared and recommended by the United States Interdepartmental Committee on Paint Specification Standardization, published in Circular 91 of the Bureau of Standards, states that the dry pigment shall be a hydrated oxide of iron permeating a siliceous base, and shall be free from added impurities. It must all pass a 200-mesh screen, contain no less than 17 per cent iron oxide, not more than 5 per cent lime, - no lead chromate or organic colors, and must equal the sample mutually agreed on by buyer and seller in color, color strength, and tone.

Ocher in paste form shall contain not more than 71 or less than 69 per cent pigment, not more than 31 or less than 29 per cent linseed oil, not more than 0.5 per cent moisture, or more than 0.5 per cent coarse particles and skins (total residue on No. 200 screen).

SUBSTITUTES

Ocher has no substitutes in the sense of similar cheaper products, since it is the cheapest of the common yellow pigments and competes, especially in the golden grades, with more expensive products like chrome and zinc yellows. Artificial ochers, of composition somewhat similar to that of natural ocher, and yellow clay mixed with more or less ocher, are used to a limited extent in the cheaper grades of linoleum. Powdered slate and ground shale are now actually preferred by certain linoleum and oil cloth manufacturers who in past years were heavy consumers of ocher.

ARTIFICIAL OCHER

Since ocher consists essentially of hydrated oxide of iron, similar material can be prepared by precipitating iron salts. Artificial ochers have been made from copperas ($FeSO_4.7H_2O$) by pouring a solution of this salt into milk of lime and thereby forming a precipitate of calcium sulphate and iron (ferrous) hydroxide. This material is shoveled onto boards, allowed slowly to oxidize, and dried promptly when the proper shade of color is reached. An artificial ocher prepared in this way is likely to contain an excess of lime, which is detrimental for many purposes. Artificial ocher suitable for calcining to form red ochers and various iron paints is prepared by chemical precipitation, often using very pure materials. Alum sludge and various metallurgical by-products containing iron salts are likewise worked up into ocher.

MINING

As a general rule, ocher is a decomposed product or iron-bearing mineral which has been transported by water (often in admixture with clay) and finally deposited in seams, pockets, or what at one time were shallow pools. Occasionally, as in Georgia, it is found just above bedrock and beneath a residual capping -- sometimes even it extends downward into cracks and fissures of the bedrock; more often, however, it forms irregular masses or lenses in residual or transported clays.

Most of the ocher deposits in the United States can not be mined systematically because of their irregular and pockety nature. Wherever possible, the overburden is stripped off, but when it is too thick, either tunnels are driven into the hillside or shafts are sunk, depending upon the topography. Shorter drifts or tunnels are driven from the main openings at suitable points, and the ocher is mined wherever it is found. A number of levels, one above the other, have been opened at several localities.

Even when the deposits are quite dry, timber ordinarily has to be used in the main drifts. Moreover, if either the ocher or the overburden is at all wet, timbering must be kept close to the face, and it will be necessary to mine out a given block of ground as rapidly as possible before too much weight comes on the timbers or before the wet clay or ocher oozes excessively into the workings. In Pennsylvania, where the deposits are erratic and the pockets of ore are

relatively small, operations are at present all open-cut, although shafts are at times employed wherever the beds lie more than 10 to 15 feet below the surface. The ocher, since it is more or less mixed with clay, must be removed very carefully; otherwise, since the clay can not be removed by washing and settling, the product becomes too low-grade.

The mining methods employed in Georgia, since they are conducted on a more extensive scale and are adapted peculiarly to local conditions, are described separately, as are also the methods used in France.

BENEFICIATION

The preparation of ocher for market is ordinarily quite simple, involving merely a rough separation of the ocher from impurities that may be mixed with it. The sequence of processes is essentially similar in all districts and includes washing, drying, pulverizing, and packing. The purpose of washing is primarily to rid the product of sand, small stones, and other foreign matter which may be associated with the ocher in nature or which has been mixed with it in mining. The washing process, however, since it thoroughly mixes the ocher, enables a more uniform product to be prepared. Sometimes it is also possible to improve the color or to produce minor modifications in the shade.

In Georgia, due to the fact that several of the deposits are large enough to justify the requisite expense, the plants are permanent and relatively more elaborate than in Pennsylvania, for example, where the deposits are small. As delivered to the washer, the Georgia ore contains 30 to 40 per cent ocher, mixed with quartzite fragments, clay, sand, and a little barite. Large rock fragments are rejected on a grizzly, and water under pressure is used to disintegrate the material and wash off ocher adhering to coarse rock. The ore then passes to a single 26-foot log washer. The logs are of yellow-pine, 18 inches in diameter, with chilled iron flights attached with lag screws. They are given a slope of 1 inch per foot, and are rotated at the rate of 20 revolutions per minute. As the coarse discharge leaves the log it is raked over a fine (about 12 mesh) screen and passes under water sprays to remove the last of the ocher before it is sent to waste. The fines through this screen, after being roughly classified in settling boxes, join the overflow from the log. The ocher, largely in suspension, can then be passed slowly through about 150 feet of wooden launders approximately 10 inches square and with a slope of one-quarter inch in 16 feet. The sand not eliminated in the log settles out in these troughs and is continuously shoveled out by hand and thrown away. In place of the launders, a classifier may be used. The washer has a capacity of 2 to 3 tons per hour.

The operation of the plants differs considerably in detail from this point on. The Georgia Peruvian Co., realizing that it was losing a large quantity of ocher by the old method of separating the sand in settling troughs, installed three James slime separatorsor classifiers in a plant located midway between the washer and the finishing plant.[4]

4 - Hubbell, A. H., An Improved Way of Washing Ocher. Eng. and Min. Jour.-Press, vol. 118, No. 9, Aug. 30, 1924, p. 336.

The entire mine output except the coarse discharge of the washer passes through the separators which recover 96 to 98 per cent of the ocher, separating all sand and other material over 300-mesh. Each of the three machines will successfully discharge (as overflow) from two to three tons per hour of minus - 300 - mesh ocher. At times, as high as four tons per hour has been discharged.

These separators, which were designed especially for this particular operation, consist of cones 14 feet in diameter. Material coming from the log washer is fed into the center of the cone by means of a launder. The ocher and slime overflow into a peripheral launder 12 inches wide. The coarser sand settles against a rising current of water that enters the cone at the bottom and is supplied from an overhead tank. The velocity of this upward current is carefully adjusted so as to effect the proper separation of sand and slime. The sand discharges through the spigot at the bottom into a small horizontal cylinder containing a flight conveyor which is operated by a 3-hp. motor mounted on top of the cylinder. This cylinder is an integral part of the cone. The flight conveyor stirs and scrubs the sand, releasing the particles of ocher that may be held between the coarser material; the ocher thus freed is pumped back into the top of the cone by a small centrifugal pump mounted on andintegral with the conveyor-cylinder. The sand tailings receive a very thorough scrubbing in fresh water, which is introduced into the discharge end of the cylinder through a second pipe line from an overhead supply tank. The flight conveyor rotates at 18 r.p.m. and has a double set of flights, so that the sand is moved very rapidly through the cylinder and is finally discharged as waste.

Before the slime separators were installed, the ore pulp after passing through the settling troughs to get rid of the sand was delivered to a 10 by 26 foot Dorr thickener preparatory to going to the drier. A very much larger volume of water, however, is now required to effect the thorough scrubbing given the sand in the separators, so that it has become necessary to install another and larger thickener. The small thickener can handle a monthly tonnage equivalent to a little over 500 tons of finished ocher. The capacity of the new thickener, which measures 8 by 34 feet, is roughly estimated at 600 to 700 tons of finished ocher monthly. The pulp entering the thickeners contain about 30 per cent solids, and the spigot discharge contains about 60 per cent. The pulp is pumped to the thickeners from the separators by a centrifugal pump driven by a 25-hp. motor.

The thickened product of the Dorr tanks is pumped by a 5-hp. motor Dorrco pump to two agitator tanks, 8 by 10 feet and 6 by 10 feet, respectively, which are homemade. From these the thickened ocher passes through $1\frac{1}{2}$ inch pipe to a so-called sprayer box at each drum drier, 10 by 12 inches in cross section, through which box a 3-inch shaft, studded radially with 3-inch pins or bolts, runs longitudinally. The shaft rotates at 200 r.p.m. in the pulp, and the pins spatter the ocher on to the slowly revolving surface of the drum. The drum is 10 feet long and 4 feet in diameter. Live steam at 80 pounds pressure is introduced into the center of one end of the drum, and the condensed water falls to the bottom where the pressure within the drum expels it via a stationary pipe bent to form a siphon which passes out of the drum through the center of one

end and thence to the boiler via an ordinary steam trap. These driers have been developed in the Cartersville district and are surprisingly efficient in view of the amount of water evaporated per pound of steam.

The ocher is in contact with the hot surface of the drum only long enough to be thoroughly dried. It is scraped off at another point before it has a chance to darken through the loss of any of its water crystallization. Darkening is to be avoided, as it spoils the color and thus lowers the grade and value for marketing. There are five of these driers, each of which has a capacity for 24 hours' continuous operation of about 11 short tons.

At another plant the ocher-laden water from the washer and settling launder is run into a series of settling tanks. After standing, the excess water is siphoned off and the thickened ocher is shoveled into wheelbarrows and transferred to two long rows of drying vats, arranged along each side of a horizontal belt conveyor. The bottoms of these vats, which are 12 to 15 feet square by 3 feet deep, are covered with coils made of 1-inch pipe through which live steam is passed when the vat is filled with ocher sludge. When the ocher is thoroughly dry it is shoveled by hand into the conveyor. A disintegrator breaks up the large lumps, and after going through a pulverizer, the ocher is taken by screw conveyor and elevator to a sheet-steel bin over the packing floor.

In Pennsylvania, the ore as it goes to the washer is mixed with clay and with nodules and fragments of limonite and chert. The clay, of course, accompanies the ocher in the overflow from the log washers. The better part of the iron ore is separated from the chert by hand, and when enough of it accumulates it is shipped to some nearby furnace.

The ocher and clay are washed into a series of settling troughs that are slightly inclined so that the water passes through them rather slowly. The current is further retarded by baffle boards, behind which the coarser particles settle. At one mine there are 28 of these troughs each 14 to 16 feet long and 13 inches wide. The coarse sand settles mostly in the first two or three troughs, and by the time the last one is reached even the extremely fine sand has been practically eliminated. From the sand troughs the ocher-bearing water flows to settling ponds formed by digging into the ground a few feet and placing the excavated material as an embankment around the sides. These ponds which are roughly rectangular in shape, vary in size, but they average probably about 40 feet in length, 25 feet in width, and 3 to 4 feet in depth. Frequently they are arranged in series so that the finest material will pass from the one pond into the next; the overflow from the last pond is carried off through a pipe. It is also possible to grade the material as it comes from the mine and then to turn the best grade of ocher into one pond and to send that having a large admixture of clay into another. When a pond is full, the surplus water is allowed to evaporate. This may require from a few weeks to several months, depending upon the weather. When the ocher finally reaches a condition where it can be readily shoveled, it is dug and hauled in wheelbarrows to the drying sheds and placed on long open shelves for final drying. In a few places steam drying sheds are used, but most of the ocher is air-dried. Some of the plants which have steam dryers use them only in winter when air-drying is impossible.

After drying, the material either is hauled at once to the railroad for shipment or else is ground in buhr mills and then packed in bags or barrels.

At the old iron mines ocher was deposited behind the mud dams constructed to impound and clarify the muddy waters from the mines and washeries. Much of this ocher is mined with sand, but some of it even without washing is almost as good as that which has been carefully washed from natural deposits. As a rule the ocher from these old mud dam deposits is apt to contain a fairly large proportion of clay and sand and must be washed, but in the extensive deposits that accumulated about the larger mines that were worked for many years it is usually possible to find several layers of fairly clean ocher.

DOMESTIC DEPOSITS

Ocher is produced more or less regularly in the following States named in order of importance: Georgia, Pennsylvania, Virginia, Alabama, California, Iowa, and Vermont. Deposits have been found in other States, but are not known to have maintained any important production. In the Pacific Northwest, however, certain deposits, notably in western Washington, are of considerable potential importance commercially.

Georgia[5]

Georgia is by far the leading producing State. There are no official figures of production, but according to estimates of several producers, the annual output is in the neighborhood of 15,000 tons, practically all of which comes from the Cartersville district, in Bartow County.

Early Developments

In 1877 E. H. Woodward began mining ocher on a property located near the limits of the town of Cartersville.[6] The crude ocher was hauled in wagons to Cartersville where it was prepared for market. At the same time Mr. Woodward was engaged in mining manganese on the Dobbins property some miles away. In 1878 A. P. Silva, another manganese producer, commenced to mine ocher in a small way. In the same year M. F. Pritchett purchased the ocher interests of both Woodward and Silva. For drying the ocher he employed a brick furnace about 30 feet long and 4 feet wide, with thin sheet iron for the bottom and a fire box located at one end.

Pritchett sold his interest to Maltby and Jones in 1879. Improved methods of mining were introduced, but the crude material was still hauled to Cartersville to be prepared for market. The only mines worked were located on the Larey property near the bridge across the Etowah River, $2\frac{1}{2}$ miles southeast of Cartersville.

5 - Weigel, W. M., Barite and Ocher in the Cartersville, Ga., District. Repts. of Investigations, Serial 2477, 1923, 11 pp.
6 - Watson, T. L., A Preliminary Report of the Ocher Deposits of Georgia, Geol. Survey of Ga., 1906, p. 67.

In 1880 this property was again sold to the Georgia Peruvian Ocher Co., which is still operating. Better methods were devised for preparing the ocher for market, and since the road was better to Emerson than to Cartersville, the plant was moved to the former place, 2 miles south of the mines. This company is credited with having sent a consignment of 50 tons of its ocher to England in December, 1890. This is believed to have been the first shipment of American ocher to Europe.[7]

Systematic mining and the use of modern machinery for preparing the material seem to date from the year 1891, when J. C. Oram of Vermont and E. P. Earle of New York became interested in the company. Both of these men had handled ocher for years in the North, and as a result of their efforts the ocher industry became firmly established in Bartow County, Ga. Instead of drying the ocher in large vats with steam pipes at the bottom, Mr. Oram simply led the material into pits dug in the ground, and allowed it to dry naturally. One of these pits would contain a carload or more of ocher when dry.

In 1893 W. B. Shaffer bought an adjoining property situated on the Emerson road, directly at the bridge across the Etowah River, and organized the Standard Peruvian Ocher Co. Three years later G. Linderman purchased the Shaffer plant and property, becoming owner of both the Shaffer and Oram mines and mill. These properties are operated at present under the name of the Georgia Peruvian Ocher Co. All the machinery and other equipment were moved from Emerson to the present site at the bridge, and a modern plant was installed at the mines for preparing the ocher for market.

A second ocher plant was erected in 1898 by the Cherokee Ocher and Barite Co., 1 mile east of the railroad station at Cartersville. This plant is in operation to-day under the same name.

In 1899 a third plant known as the Blue Ridge Ocher Co. was located about 2½ miles east of Cartersville. The last and fourth plant established in the district was that of the American Ocher Co., in 1902. The plant is located 2½ miles nearly due east from Cartersville, and like the others is in all respects an up-to-date plant.

Mining Practice

The ocher occurs along a belt extending for 6 or 8 miles in a nearly north and south direction and lying within about 1¼ miles east of Cartersville; the occurrences begin at a point about 2 miles south of the Etowah River where this stream cuts across the belt of Weisner quartzite.

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7 - U. S. Geological Survey, Mineral Resources of the United States, 1889-1890, p. 509.

The ocher in general forms an irregular network of veins in the quartzite, though many of these are neither rich enough nor large enough to work. Formerly the ocher was mined largely from open cuts near the base of the hill, but these had to be abandoned when the overburden became excessive, and now all mining is carried on underground. The vein or bed being worked strikes approximately northeast and southwest, parallelling the hillside, and as far as developed the dip (to the northwest) conforms to some extent to the slope of the hill. Toward the top of the hill the ore body flattens, follows the slope approximately down the hill, and flattens again at the foot, apparently dipping under the river in a northwesterly direction. The greatest dip on the hillside is about 40°. That part of the vein good enough to work is 12 to 40 feet thick; it is locally called sandstone, but is altered to some extent; the overburden consists almost entirely of decomposed quartzite and clay. The vein is reached by tunnels driven through the overburden at vertical intervals of 50 feet. On reaching the ore, drifts are carried along the foot wall and raises are put up about every 50 feet. The ore is stoped underhand, beginning at the top of a raise and working down the dip. At the bottom of each raise a loading chute is provided for the mine cars. The raise is never quite filled with ore. All available waste is stowed in the square sets to help hold the ground, which is very heavy and requires timbering close up to the face. Tunnels and drifts are timbered with sets 4 feet apart. Round timbers 8 to 10 inches in diameter, hand flattened, are used. As the timbers last only two to three years, all ore recoverable from one tunnel and set of drifts is removed if possible within this period in order to avoid re-timbering. On account of the short distance from the surface to the ore, it is cheaper to work different parts of the vein through a number of openings than to keep a fewer number of main haulageways and permanent tunnels in repair. One-ton ore cars on 18-inch gauge track are used, and trammed by hand and mules.

The other mines in the district are worked in much the same manner.

Production

The quantity and value of the ocher produced in Georgia during the period 1889 to 1914 is shown in the following table:

Production of Ocher in Georgia, 1889 - 1914[1]

Year	Quantity, short tons	Value	Year	Quantity, short tons	Value
1889	2,512	$29,720	1902	3,688	$38,423
1890	800	12,800	1903	5,212	47,908
1891	600	9,000	1904	4,752	44,142
1892	1,743	26,800	1905	4,209	43,481
1893	2,000	39,000	1906	5,550	58,350
1894	1,690	17,840	1907	5,600	57,100
1895	2,105	31,080	1908	6,035	63,851
1896	2,981	28,005	1909	5,838	60,971
1897	2,608	36,600	1910	7,011	70,388
1898	2,858	30,798	1911	7,395	69,447
1899	3,212	39,505	1912	10,107	101,790
1900	6,828	73,172	1913	11,420	123,090
1901	5,077	49,176	1914	8,607	84,193

1/ Later figures not available, although estimates place present output at about 15,000 tons annually.

Source: 1889-1902 Watson, T. L., Ocher Deposits of Georgia. Geol. Sur. of Ga. Bull. 13, pp. 69, 1906, 1902-1914 "Mineral Paints" Mineral Resources of the U. S. Annual U.S. Geol. Survey.

Pennsylvania[8]

Pennsylvania's annual output is estimated at about 4,500 tons. Yellow ocher occurs at many places in the State, though it has been worked mainly in the eastern part. Its distribution is practically coextensive with the brown (limonite) iron ores, which occur mostly in the limestone that crosses the State in broad or narrow bands in a general northeast-southwest direction. In recent years these iron ores have been largely neglected, but at one time they were extensively utilized.

The most important ocher district in Pennsylvania at present lies within a comparatively narrow belt of limestone and quartzite which extends from Easton to Reading, and which lies between the gneiss ridges of South Mountain on one side and the slates of the Hudson River Series on the other. It comprises the Easton, Allentown, Slatington, Boyertown, and Reading quadrangles of the United States Geological Survey, and lies in the counties of Northampton, Lehigh, and Berks.

8 - Miller, B. L., The Mineral Pigments of Pennsylvania. Rept. Topo. and Geol. Survey Com. Pa., 4th ser., 1911.

The ocher and the brown iron ores of the region occur either in pockets irregularly distributed throughout clays, or in rather definite layers, perhaps representing the strata of the original rocks that have been wholly replaced. Some of the pockets of ocher are several feet in diameter, and it is possible to remove the material without taking out much clay. In other cases, however, the ocher is in such small masses that much clay must be removed, and the mixture forms a low-grade ocher. The clay ranges in color from white to yellow, to red, to black, and since it can not be removed from the ocher by washing and settling, the best grades of ocher can be obtained only from large pockets or from thick layers.

The ochers and associated clays lie upon the older rocks, in the main, and represent the residual insoluble material or replacements that have occurred along fractures or faulted zones.

The origin of the limonite and ocher deposits of Pennsylvania has been discussed by many writers and many different explanations have been advanced. H. D. Rogers[9] believed that the iron came from the overlying slates in which the iron existed in the form of pyrite. Prime[10] says they have been formed in place by the decomposition of ferrous silicates or ferrous carbonate originally present in the limestones. The assertion is made by d'Invilliers[11] that they are produced by the decomposition of pyrite originally present in the shaly strata intercalcated with the limestone. Hopkins[12] says "the original source of the iron is primarily the Cambro-Ordovician limestone and slates, with smaller quantities from the overlying Ordovician and possibly Silurian strata and the underlying slates and quartzites. The iron occurs in these strata as carbonate, sulphide and silicate, the first being probably the most common."

The deposits lie mostly between Easton and Allentown and the mines are served by the Lehigh Valley Railroad Co. and Central of New Jersey, those mines lying between Allentown and Reading are served by the East Pennsylvania branch of the Philadelphia and Reading Railway Co. which follows closely the line of ocher working. Throughout the belt most of the ocher mines are within 3 miles of the railroad, so that the cost of haulage to the shipping points is not excessive as the roads are generally good. Steam power is commonly used.

Production

The quantity and value of the ocher produced in Pennsylvania from 1889 to 1914 is shown in the following table:

9 - Geology of Pennsylvania, vol. 1, 1858, pg. 183.
10 - Second Geological Survey of Pennsylvania, Rept. D., 1874, pp. 53 and 59.
11 - Second Geological Survey of Pennsylvania, Rept. I, p. 36.
12 - Bull. Geol. Soc. Amer., Cambro Silurian Limonite Ores of Pennsylvania. Vol. 11, 1900, pp. 475-502.

Production of Ocher in Pennsylvania[1]

Year	Quantity, short tons	Value	Year	Quantity, short tons	Value
1889	7,922	$103,797	1902	9,818	$80,259
1890	4,173	61,458	1903	4,937	34,782
1891	4,535	56,588	1904	4,077	29,355
1892	7,055	90,755	1905	7,789	72,360
1893	5,375	71,575	1906	8,597	79,244
1894	4,975	47,830	1907	8,047	76,816
1895	6,800	74,300	1908	9,286	78,956
1896	2,926	26,818	1909	4,137	45,472
1897	6,825	81,325	1910	3,642	32,254
1898	5,986	61,500	1911	3,013	28,101
1899	7,285	57,245	1912	3,300	28,950
1900	7,601	21,661	1913	3,935	32,175
1901	7,632	76,106	1914	3,799	34,223

1/ Official figures after 1914 are not available.

Source: "Mineral Paints," Mineral Resources of the U. S. Annual, U. S. Geological Survey.

Virginia

Little ocher has been produced in Virginia in recent years, although a few tons may have been shipped without finding their way into the recorded figures of mineral production of the State.

Ocher of more or less purity is found and has been produced to some extent in each of the principal geologic divisions of the State, namely, the Coastal Plain, the Piedmont Plateau, and the Valley region. At one time or another it has been mined in the following localities: Near Bermuda Hundred, on the Appomattox River, in the extreme eastern part of Chesterfield County; in the Little Catoctin Mountain, near Leesburg in Loudoun County; near Bedford City in Bedford County; near Keezletown in Rockingham County; from the western base of the Southwest Massanutten Mountain; at Stanley in Page County; and near Shenandoah in Page and Rockingham Counties. Other ocher deposits are found rather widely distributed over the Valley and Piedmont provinces and to some extent over the Coastal Plain, but they have not been worked. Deposits of ocher varying in color from red to yellow and brown, some of them apparently quite promising, are found in Campbell and Bedford Counties; near Bon Air in Chesterfield County; near Fairfield in Rockbridge County; near Waynesboro in Augusta County; and near Roaring Run in Craig County. In the Valley and Piedmont areas, the ocher beds are usually associated more or less with beds of iron ore.

The writer has received several fine specimens of yellow ocher from the vicinity of Bluemont, in Loudoun County, and W. P. Miller, of Lynchburg, Campbell County, has sent in a specimen of bronze ocher which no doubt might be used for a paint pigment.

Other States

In Vermont, ocher mining is one of the oldest minor industries. Beds at Brandon in northern Rutland County and at Shraftsbury and Bennington, in Bennington County, have been worked in a small way for many years, but is is not known whether at present the ocher mines of the State are producing or not.

In Alabama, ocher is mined in Clarke County, and in the past has been produced in Autauga and Elmore Counties.

In California, ocher deposits have been worked in Calaveras, Napa, and Riverside Counties.

STATUS OF THE DOMESTIC INDUSTRY

The ocher industry in the United States comprises mostly small intermittent operations, although some permanence is in evidence in the Georgia field. Owing partly to the pockety nature of the occurrence and partly to the difficulty and uncertainty of finding a continuous market, domestic ocher suffers severely in competition with the well-established French products which are considered superior to the domestic product. Imported ochers, as a class, have a better permanent color than those produced in this country, but their leading advantages are the result of more uniform deliveries and standardized grading. In the linoleum and oilcloth industries, into which the domestic product enters in large quantity, exact shades of color are of less significance than other properties.

No actual production figures have been published since 1914. In that year 14,387 tons were produced, having a value of $136,185, which were divided as follows: Georgia, 8,607 tons, valued at $84,193; Pennsylvania, 3,799 tons, valued at $34,223; and other States 1,981 tons, valued at $17,769.

In the reports of the Bureau of the Census, the production of ocher is included with that of other iron oxides, both natural (sienna, umber, etc.) and synthetic (precipitated). For this larger group the output was reported as 54,180 tons valued at $3,357,895 in 1927, a large increase as compared with 33,895 tons, valued at $2,151,445, reported for 1925. For ocher alone, unofficial estimates place production at approximately 22,000 tons, proportioned as follows: Georgia, 15,000 tons; Pennsylvania, 5,000 tons; and all other States, 2,000 tons. There is no doubt, however, that some yellow clay is included in these estimates, especially in Pennsylvania.

IMPORTS AND EXPORTS

The imports of ochers and siennas into the United States in recent years have shown a gradual increase, as will be noted from the tables following. The tonnage from France, which furnishes the bulk of the imports into the United States, has shown practically no increase, but the value of the ocher imported from that country has more than doubled during the past four years.

Ochers and Siennas Imported Into the United States, 1924 - 1927

Country	1924		1925		1926		1927	
	Pounds	Value	Pounds	Value	Pounds	Value	Pounds	Value
France	16,360,765	$161,461	17,151,162	$191,182	16,148,144	$266,176	16,658,319	$323,601
Germany	363,849	6,952	161,819	2,772	336,871	6,611	254,114	5,531
United Kingdom	190,913	5,773	245,298	9,293	219,552	7,388	321,469	15,230
All Other	2,741,760	75,497	2,580,535	71,773	4,141,129	109,976	3,638,467	104,132
TOTAL	19,657,287	249,689	20,138,814	275,020	20,845,696	390,151	20,872,369	448,494

Ochers and Siennas, Unground Crude, Imported for Consumption

Year	Ochers		Siennas	
	Pounds	Value	Pounds	Value
1924	328,912	$5,088	1,501,490	$42,473
1925	134,775	2,214	1,385,608	34,809
1926	122,697	2,717	2,018,289	49,472
1927	466,732	13,107	1,668,065	42,831

Ochers and Siennas, Washed or Ground, Imported for Consumption

Year	Pounds	Value
1924	17,994,225	$204,237
1925	18,651,896	238,258
1926	18,251,765	334,693
1927	18,920,745	395,684

THE FRENCH INDUSTRY

General Statement

The principal ocher deposits of France are located at two main centers: At Apt, in the Department of Vaucluse, and at Auxerre, in the Department of Yonne.[13]

The ocher deposits of the Department of Vaucluse are located in two valleys of the Provence Alps; that of Calavon Torrent between the Luberon and Vaucluse Mountain ranges, and that of the Auzon River between the Vaucluse and Ventoux ranges.

The physical properties of the ore in this district are favorable to low production costs and high quality of product; the material is reported as being superior to that of any other country in the world. The ocher content of the ore extracted at present ranges from 5 to 20 per cent; a good average is said to be 10 to 15 per cent. A full color range exists - from light "canary yellow" to the deepest reds. The product is also characterized by impalpable fineness, without appreciable loss of color in grinding.

The Gargas (Vaucluse) field is known to be the world's greatest producer of "canary" and "lemon" (citron) yellow ochers, of "French satin" grade. Its product is in great demand and established factories compete in buying it. The exploitation in this field is carried on by farmers and by two of the leading

13 - Cameron, A. E., Ocher Industry in the Marseille Consular District, France, 1923, pg. 1.

members of the "Comptoir," subsequently described. These two companies own their own land and also the mills for treating the product. The farmers, on the other hand, sell their ore to the highest bidders. Those possessing good ore and sufficient water, however, find it to their advantage to wash the ore themselves and haul only the washed product to the Apt market.

For the farmers in this region, mining is a seasonal occupation; each farmer, to some extent, suits his own convenience as to the season for digging, storing the ore, and washing it, if he is a producer of washed ore.

The ore is usually dug during the fall and early winter and stored in separate lots according to color. It is generally washed during the rainy season, January to April, and dried in covered sheds from April to July. Further manufacture then continues during the rest of the year.

An interesting method of quarrying is that developed by the Eugene Dagan & Cie. of Apt, at a deposit at Roussilon. Here approximately horizontal adits are driven into the hillside and are well timbered. They follow particularly rich veins and pockets to a considerable depth, and permit the choice of good quality ore before extraction. The extension of these tunnels into other privately owned land is practically impossible, as the royalties demanded exceed what the companies can afford to offer under present conditions.

In preparing the material for market, ore of as nearly uniform color as possible is selected, dumped into a basin, flooded, and allowed to settle for several months. These basins are from 9 to 15 feet square and are made by damming small streams. After the mud cake is formed the water is drained off, and the cake is taken out in sections and placed in covered sheds to dry. It is then pulverized by hand with a large pestle, and put through fine mesh screens. This product is the "washed" ocher of commerce, marked with the letter "L" (lavee or washed), as distinguished from the washed, ground, and screened ocher of the factories. Opinions in the trade differ as to the relative merits of the old-fashioned farm product and the newer factory product.

If water is not available, the owner of the ore sells it to the factory offering the best price, and often delivers it to the plant himself.

In preparing ocher for the market in plants, the time-honored methods and routine of the farmers are employed, but with several improvements. The vats, or basins, are built of concrete and are larger and deeper, averaging 45 by 60 feet. Several basins are in operation at each plant, permitting several colors to be worked at the same time and a finer assortment to be obtained in each color. Grinding, cleaning, bolting, sorting, and grading are carried out mechanically. Centrifugal tables revolving at high speed sort out impurities with great accuracy and economy.

Day labor at the plants is employed at current rates, from 15 to 18 francs (60 to 75 cents) per day for unskilled labor. The labor supply is abundant, and accommodates itself remarkably well to seasonable migration between the varied industries of the region; farming, canning, preserving, and the

mining and preparation of ocher. High labor costs are kept down by gradually increasing the percentage of unskilled laborers as the division of labor becomes possible. The feeding of the grinding machinery and the packing of the product in barrels are now unskilled tasks.

The plant motive power in general use is steam, but electricity is being used more and more, as it is more economical. Narrow-gauge track exists in most plants, permitting the change from horse to electric locomotives.

Production

The production of ground and washed ochers in the Department of Vaucluse in recent years has been in the neighborhood of 23,000 metric tons, about two-thirds of the production of the pre-war years. The United States imports about one-third of this amount annually, or 8,000 tons of washed ocher.

Production of Ocher in France by Departments
(Metric tons)

Department	1913	1921	1922	1923	1924	1925
Ardeche	120	- -	650		- -	- -
Ardennes	1,000	- -	- -		- -	- -
Drome	120	1,000	1,000		1,000	850
Gard	- -	- -	- -		- -	- -
Pas-de-Calais	- -	500	150		- -	- -
Pyrenees (Hautes)	- -	- -	- -		500	600
Vaucluse	35,000	10,000	14,000		22,000	23,530
Yonne	20,000	7,000	11,960		13,000	- -
Nievre	- -	200	- -		- -	- -
Total	56,240	18,700	27,760	26,700	36,500	24,980

Organization of the French Industry

In an industry whose market is world-wide and whose most efficient producing region is extremely small, it is not surprising to find evidence of various attempts to control production and distribution by means of private combination.

The earliest manifestation of the natural tendency toward combination was the formation of a selling cartel known as the Chambre Syndicale des Fabricants d'Ocres, an all-embracing and powerful syndicate with many individual members, each of whom produced or traded on a small scale. Embracing both sellers and buyers, it formed a clearing house and a place for the discussion of costs, prices, and selling policies. Its weakness was twofold - inability to hold its membership in line during the lean war years, and the gradual ascendancy of one of its members, the Societe des Ocres de France, long the leading single ocher producer of France. After the formation of the Comptoir, which took over some of the Syndicate's most prominent local members, the need for the Syndicate was at an end. No evidence of its existence has been found since 1922.

The next step in the progress of the industry in Vaucluse was the entry into the Vaucluse district, as a producer and manufacturer, of the Societe des Ocre de France. This company, organized more than 60 years ago at Lyon, had operated very successfully in the Auxerre (Yonne) region, and had always enjoyed an excellent reputation for the color, texture, and purity which it extracted from the comparatively lean ore of the Yonne fields.

This concern, after entering the Apt district and operating as buyer and seller of ochers, built a modern plant at Apt. As it is located on the banks of the Calavon Torrent, it has ample water supply for its large washing pits or basins. Its steam-powered grinding mill treats mainly ore sold to it by many of the smaller land owners of the district. When opportunity arises it uses ore from its own extensive properties.

The remarkable success attained by the Societe, based on improved production methods and by large scale operations at more than one point, was felt by many of its competitors as a constant menace. This feeling was accentuated by the knowledge that the Societe, even before establishing its factory at Apt, was already the largest single factor in the French ocher industry. The smaller plants and dealers either had to combine or resign themselves to selling to or through the Societe.

A new combination was inevitable, and was made up of a few of the independents. It took the form of a centralized selling organization, and was called the "Comptoir des Ocres Francaises," a company with varying membership and capital, each member having one share -- a convenient form of cooperation open to both individuals and corporations under the French law. Each member retains actual ownership of his own ocher deposit and grinding mill, and thus continues to represent an integral production unit whose internal organization and production policies are not interferred with. Sales, on the other hand, are absolutely in the hands of the Comptoir, and no member is permitted to sell even to another member of the cartel. Each member has a representative on the board of directors, and all members sit in the board meetings.

The present membership of the Comptoir includes:

Eugene Dagan & Cie.,	Apt.
Faustin Caste,	Apt.
Leopold Anseline,	Apt.
Julian Freres,	Apt.
Francis Barthelemy,	Apt.
Ad. Jean & Cie.,	Apt.
Tamisier Freres, Gargas	Vaucluse.
Aug. Malavard Fils,	Villes-sur-Auzon.

The determination of a selling policy was the principal difficulty in the launching of the Comptoir. So many of its members had exclusive agents in certain foreign countries that it was impossible to confine representation of the Comptoir to any one agent in any country. The policy of exclusive agents

was therefore abandoned by common consent. However, the former agents of each of the individual producers are now the favored customers of the Comptoir, and have protected distribution in the countries in which they operate.

Production, sales, and exports of French ocher are almost entirely in the hands of the Societe des Ocres de France and the Comptoir des Ocres Francaises. They compete on equal and friendly terms and so successfully as to control the market for French satin ocher both in France and abroad.

Grades of Ocher

The basic grades of commercial ocher on the French market are marked as follows:

J. L. - Washed yellow.
J. C. L. - Washed light yellow.
J. F. L. - Washed dark yellow.
CITRON - Lemon yellow.
R. L. - Washed red.
R. F. L. - Washed dark red.

If the texture of the product justifies the name, the letter S (satin) is added, representing a considerably increased value. Similarly, finely distinguished colors are designated by appropriate letters. All letters are branded or stamped on the heads of the barrels.

Six-letter satin ocher usually commands from four to six times the price of two-letter basic grades, when furnished by a reputable producer or dealer. Formerly many farmers were adept in producing the six-letter grades of remarkably uniform texture and color, and the trade was handed down from father to son. Now most of the finer grades can be made in plants under supervision of experts, and the plants also dominate in the production of light colors - canary, lemon and light red - with which iron oxides and other substitutes can not compete.

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DEPARTMENT OF COMMERCE -- BUREAU OF MINES

GEOPHYSICAL ABSTRACTS

NO. 11

BY

FREDERICK W. LEE

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GEOPHYSICAL ABSTRACTS [1]

No. 2

Compiled by Frederick W. Lee [2]

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2 - Senior physicist, U. S. Bureau of Mines.

1 - GRAVITATIONAL METHODS

THE EÖTVÖS TORSION BALANCE

Anonymous (probably Capt. H. Shaw and E. Lancaster-Jones)

Published privately by L. Oertling, Ltd., as a manual to be used with their torsion balance. London, 90 pages; undated, but late in 1925 or early in 1926.

The content of the book covers: (1) The theory of the Eötvös torsion balance; derivation of the working formulas; description of the instrument; method of observation; relation of the magnitudes observed to the lines of force and the level surfaces of gravity; corrections or normal effects, near and distance terrane; subterranean effects; effects due to certain mathematical bodies. (2) Notes on the practical employment of the Eötvös torsion balance; initial testing of the instrument and of accessories; hints on planning the survey; work at the station site; office computation and plotting of results; the interpretation of the results. (3) A description of the instrument and instructions for its manipulation.

This book was the best discussion in English on the theory and use of the Eötvös torsion balance and is surpassed only by a later paper by the same authors that covers much of the same ground. The book is suitable for a textbook for beginners, and advanced students will find it of use. It should be available as a reference book in all geological libraries and in all torsion balance department offices.

The discussion is authoritative and is given as simply as it is possible to give theory that is not very simple. Formulas for the terrane correction calculated by Schweydar's method are given in terms of tenths of feet difference in elevation and may be welcome to some who desire feet throughout in the measurements. -- Donald C. Barton.

THE EÖTVÖS TORSION BALANCE METHOD OF MAPPING GEOLOGIC STRUCTURE

By Donald C. Barton

Am. Inst. Min. Eng. Tech. Pub. No. 50, February, 1928, 51 pp., 13 figs.

An excess or deficiency of mass causes a warping of the level surfaces and of the lines of the vertical. Either type of warping gives a small horizontal component to gravity at each of the two weights of an Eötvös torsion balance. That small component in one case is proportional to the differential curvature of the level surfaces and in the other case, to horizontal gradient of gravity. The formulas for the calculation of the differential curvature and of the gradient from observations with the torsion balance are derived

geometrically. The effects measured by the torsion balance are the vectorial sums of the effects produced by the topographic, the planetary, and the geologic irregularities of the distribution of mass. By running levels and using certain formulas, corrections can be applied to compensate the effects of topography of minor relief. A correction varying with latitude is applied to compensate the planetary effects. From a study of the residual geologic anomalies, it is possible to interpret certain features of the geology. The theoretical gradient and differential curvature profiles are given for a vertical and a 30° fault cutting formations of increasing density downward, of an infinite symmetrical structural ridge, and infinite asymmetrical structural ridge, a finite asymmetrical structural ridge. Illustrations of actual torsion balance work are given as follows: the survey which led to the discovery of the Nash salt dome, Tex., and the prediction of the limits of the dome and its verification by subsequent drilling; calculation of the extension of the cap rock on the Hoskins Mound salt dome and the subsequent verification by drilling; a profile across the Luling fault, a survey of a buried granite Ordovician ridge in Cooke County, Tex., and an analysis of a survey of the Fox oil fields, Oklahoma. The torsion balance has a distinct place and a good future in mapping geologic structure; in some situations it works brilliantly but is not a panacea for mapping geologic structure and can not replace but merely supplement geology. Although it is easy to train an observer, the interpreter must be a well trained and experienced geophysicist and geologist. The torsion balance offers usefulness in certain types of pure-science geologic problems. -- Author's abstract.

RESULTS OF TORSION BALANCE MEASUREMENTS IN SCHLESWIG-HOLSTEIN (ERGEBNISSE VON DREHWAAGEMESSUNGEN IN SCHLESWIG-HOLSTEIN)

By Karl Jung

Ztschr. fuer Geophysik, Jahrg., 4, Heft 7/8, 1928, pp. 395-450.

A single profile was run with a torsion balance between Husum and Flensburg across the positive magnetic anomaly mapped by Reich. The gradient values show several anomalies of much lesser size than the magnetic anomaly, but the profile of $\triangle G$ shows a very rough parallelism to the profile of $\triangle Z$. Approximate determination of depths by Jung warrant the suggestion that uplifted ancient rocks are the cause of those gradient anomalies, as Devonian is known to be practically at the surface at one place on the profile, but irregularities of the topography on the Tertiary or irregularities in the diluvium may cause such gradient anomalies. The anomalies of $\triangle G$ and $\triangle Z$ are both apparently due to some deep-seated anomalous mass. -- Donald C. Barton.

GRAVITY SURVEYING IN GREAT BRITAIN

By H. Shaw

Am. Inst. Min. and Met. Eng. Tech. Pub. 74, 1928, 14 pp. Also in Geophysical Prospecting, 1929., Am. Inst. Min. and Met. Eng., 1929, pp. 530-543.

The intensive torsion balance work in Great Britain has been in connection with the location of usually large and irregular deposits of iron ore. The situation was complicated by the presence of 0 to 250 feet of heterogeneous glacial drift, a subdrift topography with considerable relief on limestone, and numerous faults. On account of the superficial heterogeneity seriously affecting the differential curvature, it was neglected, and only the gradient was used. A station interval of 100 and in some cases of 50 feet was used. Many details of the field practice are given. The paper unfortunately is not illustrated with any of the results of the survey. -- Donald C. Barton.

COMPUTATION OF EÖTVÖS GRAVITY EFFECTS

By E. Lancaster-Jones

Am. Inst. Min. and Met. Eng. Tech. Pub. 75, 1928, 55 pp. Also in Geophysical Prospecting, 1929., Am. Inst. Min. and Met. Eng., 1929, pp. 505-559.

The paper summarizes practically all the known formulas for the calculation of Eötvös gravity effects. Many of the formulas are given in a mathematically pretty but very condensed notation which is rather forbidding to the uninitiated but which becomes clearer after a little study. The formulas are handled under the headings of formulas for point elements, line elements, terrane features, topographical features, cartographic features, three dimensional features bounded by plane surfaces, irregular three dimensional figures (Nikiforov's method), and change of axes.

This paper is a most useful compendium of formulas. -- Donald C. Barton.

A CARTOGRAPHIC CORRECTION FOR THE EÖTVÖS TORSION BALANCE

By C. A. Heiland

Am. Inst. Min. and Met. Eng. Tech. Pub. 52, 1928, 17 pp. Also in Geophysical Prospecting, 1929., Am. Inst. Min. and Met. Eng., 1929, pp. 544-560.

The "cartographic correction" is the correction for the effect of the topography beyond 100 meters from the instrument. Eötvös used a simple but approximate formula; Schweydar used more rigorous formulas. The writer divides

the terrane into zones bounded by radii and concentric circles. By assumption of certain approximations, a formula is derived which depends on certain coefficients and the square of the elevation referred to the center of gravity of the instrument as the origin. Rule-of-thumb tabular calculation forms, including the coefficients, can be set up; the squares of the elevations are entered in the proper spaces and added or subtracted and multiplied by an indicated routine. The respective calculations for U_{xz}, U_{yz}, U_{xy}, and $U_{\triangle}$ each require a separate form.

This is a valid method of calculating the cartographic correction. But this type of method of calculation of the correction is being superseded by graphical methods or methods making use of tables of effects rather than tables of squares. -- Donald C. Barton.

A NEW GRAPHICAL METHOD FOR TORSION BALANCE TOPOGRAPHIC CORRECTIONS AND INTERPRETATIONS

By C. A. Heiland

Bull. Am. Assoc. Petrol. Geol., vol. 13, No. 1, Jan., 1929, pp. 39-74.

The graphical methods suggested by Numerov, Nikiforov, Jung, Lancaster-Jones, Barton, and Haalck involve approximations which the author seeks to avoid by a graphical method making use of spherical coordinates. The space around the origin is divided into 16 equal vertical sectors. The gradient and differential curvature effects of the 16 sectors are shown by six different graphs. The construction of the graphs is described. To use the graphs the topographic or structural profiles are constructed for the 16 sectors and superimposed on the respective graphs. The effects are determined by counting up mass elements of the graph and multiplying by the respective density factors. These graphs according to the author, are applicable (1) to the computation of topographic corrections and of corrections for mine openings in underground torsion balance work, (2) to computations of the influence of two and three dimensional subterranean features in torsion-balance interpretations; and (3) to computations of the influence of such structures on the magnetometer.

This method is a valid and workable one for handling extremely complicated topographic corrections in regions of rough relief. As given, the drafting of the topographic profiles will be found too tedious for most commercial surveys. But a slight modification of the method with the use of templates and contour maps will be a valid and workable one for most commercial surveys. The method is unworkably complicated for most calculations in connection with subsurface structures. -- Donald C. Barton.

CALCULATIONS IN THE INTERPRETATION OF OBSERVATIONS WITH THE EÖTVÖS TORSION BALANCE

By Donald C. Barton

Geophysical Prospecting, 1929, Am. Inst. Min. and Met. Eng., 1929, pp. 480-504.

Success in interpretation of the observations depends in considerable part on the knowledge gained by calculating the effects produced by known bodies or by calculating the bodies which will produce a known effect. Most of the simpler geologic structural situations may be split into horizontal rectangular prisms, and the effect of the body can be calculated by calculating the effects of each prism by the ordinary well-known algebraic formulas. Very many of the structures in which the oil geologist or geophysicist is interested have, or may be assumed to have an axis of symmetry. Calculation of the effects on such an axis of symmetry may be very greatly expedited and simplified by the use of graphic charts of a type suggested by the writer. Each chart represents the cross-sections of a series of horizontal prisms at right angles to the plane of the section. Each prism extends to equal distances on either side of the plane of the section and is so constructed that it produces an effect of 1. (2 or 5) E at the origin.

By superposition of the cross section of a body on the graph and counting of the squares under the cross section, the effect of the body at the origin is calculated. A standard series of charts can be constructed with different ratios between the length and depth of the constituent prisms. Calculations of unknown bodies have their limitations; by use of this graphic method of calculation, a series of triangles with a common apex were found, all of which produced the same gradient and differential curvature profile within the accuracy of field observations. The use of this graphical method of analysis of an unknown structure is illustrated by application to the results of Schumann's survey of the Lanzendorf Dome near Vienna. This method of graphical calculation is not adapted for the ordinary terrane calculations or for calculations of effects not on axes of symmetry. -- Author's abstract.

THE USE OF THE TORSION BALANCE IN THE INVESTIGATION OF THE GEOLOGICAL STRUCTURE OF SOUTHWEST PERSIA

By W. F. P. McLintock and J. Phemister

Summary of the Progress of the Geological Survey of Great Britain for 1926. Appendix 2, London, 1927, pp. 168-196.

Two officers of the survey who through the invitation of the Anglo Persian Oil Co. were sent to Persia to investigate the geophysical work of the company have reported their findings in a paper which gives the results of three torsion balance traverses, the interpretation of the results and the success of the torsion balance in indicating the partially known geology, discusses

briefly the theory of the torsion balance and the interpretation of torsion balance surveys, and gives examples of the anomalies produced by different types of geologic structures. A selected bibliography of papers on the torsion balance is appended. It is an excellent and interesting paper. -- Donald C. Barton.

A GRAVITY SURVEY OVER THE SWYNNERTON DYKE, YARNFIELD, STAFFORDSHIRE

By W. F. McLintock and J. Phemister

The Mining Magazine, Dec., 1927, pp. 363-365.

A line of torsion balance stations was run across the supposed prolongation of a basalt dike, where the dike was supposed to be covered with glacial drift. The results show the presence of the dike with the clearness of a textbook example. -- Donald C. Barton.

THE DETERMINATION OF THE POSITION AND EXTENT OF SIMPLE BODIES BY MEANS OF THE USE OF THE GRADIENT AND DIFFERENTIAL CURVATURE VALUES

By Karl Jung

Ztschr. fuer Geophysik, Jahrg. 3, Heft 6, 1927, pp. 257-280.

Jung reports concisely the results of a mathematical study of the theory of the interpretation of the form and position and size of simple bodies from the form of their gradient and differential curvature profiles as the latter would be obtained for example by a survey with an Eötvös torsion balance. On account of the difficulty of handling the more complicated cases, the study was restricted to bodies (a) of simple cross section, - i.e., horizontal prisms with vertical or inclined front and back faces - horizontal plates with a vertical or inclined face, cylinders, spheres; (b) of infinite extent at right angles to the plane of the cross section (except for the sphere); (c) of homogeneous density in a medium of homogeneous density; (d) where the difference between the density of the body and of the surrounding medium is known. By the use of the relations of the abscissas of the points of algebraic and numeric maximum and minimum of the gradient and differential curvature, he gives formulas or graphs for the recognition of the presence of those simple forms of bodies and for the determination of their dimensions and of their depths below the surface. He gives also an example of the rough applicability and yet the danger of the use of the formulas in connection with bodies that are not infinite or approximately infinite at right angles to the plane of the section.

This most interesting paper is a most valuable contribution to the literature on the theory of the interpretation of torsion balance results. But the applicability of these many formulas will be extremely limited in practice; most geologic structures are not long enough in reference to their depth to be treated as infinite and are not simple enough to be represented by such simple

cross sections; only a few actual structural situations can be treated as being of homogeneous density in a medium of homogeneous density; the density situation usually can be guessed only very approximately; and, in general, except for structures coming almost to the surface, the abscissas of that distance vary rapidly with a slight change in the ordinates of the value of the gradient or differential curvature near the points of numerical maximum of the gradient or the differential curvature. With the usual complication of the affects of irregularities of lesser and greater order obscuring the situation, the abacissas of the algebraic maxima and minima are extremely indefinite and intangible magnitudes in practice, and the probable error in the determination of the abscissas may be several times the significant differences indicative of the different forms of the bodies and different depths. These formulas will have a very limited applicability in a very rough qualitative determination, for example, of whether a body is half a mile thick and half a mile deep or 2 miles deep and 2 miles thick, but in general will not give accurate enough results to be of practical use. The paper is one which should be studied by students of interpretation but which should be used with great caution in practice. -- Donald C. Barton.

GRAVITY SURVEYING IN GREAT BRITAIN

By H. Shaw

Am. Inst. Min. and Met. Eng., Tech. Pub. 74. Also Geophysical Prospecting, 1929. Am. Inst. Min. and Met. Eng., pp. 530-531.

Shaw gives a brief account of the field operations with the Eötvös torsion balance and the problems of interpreting the results in Great Britain. In that country, the balance is used almost entirely to locate ore deposits. These ore deposits are very small in size and many stations have to be taken from 50 to 100 feet apart in order to differentiate the effect of the ore from the effects of small faults, terrane, topography, overburden, and limestone rock. Since the ores are usually located in country where the surface is fairly irregular, great care must be taken in calculating the gravitational effects of the terrane and topography as accurately as possible. When these corrections are made, an isogam map is constructed and from it the effect of the ore deposits can be fairly well distinguished from the effect of the subsurface structure and its dimentions can be calculated with an accuracy sufficient to justify the survey. To calculate effects of these various disturbing elements, it is necessary to know the specific gravity of each.

The gradient is more valuable than the differential curvature in the search for small deposits, because it is not affected by the topographic irregularities as greatly, and therefore can be calculated more accurately with the same amount of work. -- Elizabeth Buhler.

EXPERIMENTS WITH E. T. B. IN THE TRI-STATE ZINC AND LEAD DISTRICT

By P. W. George

Am. Inst. Min. and Met. Eng. Tech. Pub. 65, 1928, 9 pp. Also Geophysical Prospecting 1929. Am. Inst. Min. and Met. Eng., 1929.

In his brief paper, George gives a detailed account of the gravitational results around Duenweg, the Federal Brewster lease, West Picher, and the western part of the Federal Jarrett lease. He gives four cross sections and several tables showing the drilling discoveries; also two gradient maps, one near Picher and another on the Jarrett lease. From these results, George concludes that under favorable conditions, which he lists, a torsion balance survey in the Tri-State district is very helpful in outlining the silicified and fractured areas, around the edges of which the richest ore bodies are found. -- Elizabeth Buhler.

A GRAVITATIONAL SURVEY OVER THE BURIED KELVIN VALLEY AT DRUMRY, NEAR GLASGOW

By W. F. P. McLintock and J. Phemister

Trans. Roy. Soc. Edinburgh, vol. 56, No. 7, Pt. 1, 1928, pp. 141-155.

With an Eötvös torsion balance the mass of rock south of Drumry was mapped and calculated to have a slope of 10° on the east side, changing to a slope of $2\frac{1}{2}^{\circ}$ moving westward, at depths corresponding to well data. From both the gravitational map and the well data, a known buried channel was outlined to run north-south on the east side of Drumry with a sharp turn to the east at about 2,000 feet north of Drumry.

In making this survey, use was made of Schweydar's method of calculating the terrane effects by the more accurate method of considering the Z^2 terms as well as the Z terms; and of Lancaster-Jones' method of evaluating topographic effects from a topographic contour map.

In calculating the types of structures causing such gradients, the various common formulas are applied. For a first rough approximation to the slope and depth of the structure application was made of Eötvös' formula using isogam intervals. A detailed description is given of Mekel's scheme of combining the derivitives of U_{xz} and U_{Δ} with respect to $\underline{x}$ with the observed gradient and curvature profiles, to solve for the depth, slope, and thickness, of the inclined edge of a horizontal plate infinite in three directions. -- Elizabeth Buhler.

THE NEW TYPE OF GRAVIOMETRIC VARIOMETER WITH SHORT PERIODS

By P. M. Nikiforov

Bull. Inst. pract. Geophys., vol. 3, Leningrad, 1927, pp. 308-316.

The large amount of time required for taking readings with the common types of torsion balances and the difficulties in transporting the instruments and outfit are a great disadvantage in their use. Besides, the Russians have repeatedly objected to the use of photographic registrations (Bull. Inst. pr. Geophys., vol. 1, Nikiforov; and vol. 2, Ghirin) and claim they can bring a number of data which show without doubt that readings taken in this manner can not be trusted.

Therefore, a new instrument was designed which is based on a slightly changed principle. This instrument also has the suspending wire, the balance beams, and the hanging weights, but it is not necessary to wait until the beams come to a rest. The position of the equilibrium is rather obtained by measuring the coordinates of the curve resulting from the movements of the beam itself. It then is sufficient to measure only two ordinates; and by not waiting until the beams have come to a rest much time is saved.

The suspending wire can be considerably shortened, and in the new Russian instrument a wire about 2 centimeters long is used. Nevertheless the sensitivity of the instrument is claimed to be about the same as that of the former types; namely, 2 x 10^{-9} cgs, which is entirely sufficient.

The readings are taken visually, and special care is taken to avoid influences that might come from the body of the observer. The moving of the instrument into another azimuth is made manually; no clock work is employed. To protect the beams against hard knocks a special device was designed. No tent is required except a small and light protecting one in bad weather. The instrument can be easily carried by one man which facilitates the moving to other stations. It is set up upon a tripod instead of a special foot.

The mathematical theory of the instrument was given in a previous paper. The present article contains only a brief description of the instrument. At its end the advantages are summarized, the most important ones of which are as follows: Observation time is considerably shortened, and, therefore, more stations can be perfected per day (20 stations per day and instrument); transportation of instrument is very easy; the short torsion wire is not only much safer against breaks, but also allows the selection of a more uniform and perfect segment; the influences of temperature can be disregarded because of the short observation time; no automatic accessories are employed, and therefore all troubles which may be created by them are avoided; the taking of visual readings also eliminates all trouble with photographic plates; no helpers are needed for carrying accessories, etc.

The new instrument seems to be a remarkable improvement on torsion balances. If the claims against damping and photographic registration are correct, then also the results should be more accurate. Besides, the considerable improvement in speed and transportability of the instrument is quite remarkable. -- E. U. von Buelow.

THE PHYSICAL PRINCIPLES OF THE GRAVITATIONAL METHOD OF PROSPECTING

By P. Nikiforov

Bull. Inst. Pract. Geophys., No. 1, Leningrad, 1925, pp. 153-259; No. 2, Leningrad, 1926, pp. 233-264.

A very detailed description, illustrated by many diagrams and tables, is given of the physical principles of the gravitational method of prospecting.

The contents of the work are: Chapter I, Description and theory of the torsion balance. Potential of gravity: (1) Application of the principle of the torsion balance for the determination of the elements of the gravitational field; (2) description of a torsion balance (Eötvös and that constructed in the Physico-Mathematic Institute of the Russian Academy of Sciences); (3) forces acting on the instrument; (4) differential equation of the proper movement; (5) the general integral of the equation; (6) the determination of the state of equilibrium of the torsion balance; (7) an example; (8) determination of the second derivatives of the potential of gravity by means of a graviometric variometer; (9) an example; (10) determination of constants; (11) an example.

Chapter 2, Geodetic and physical meaning of the second derivatives of the potential of gravity. Normal values, local effects and perturbations. (1) Some properties of the potential of gravity; (2) gradients of gravity along the horizontal coordinate axes; (3) changes of g; (4) the deviation of the level surface from the spheric form and the azimuths of the principal normal sections; (5) normal values, local effects, and perturbations of the second derivatives of the potential of gravity; (6) normal values (tables with values for latitudes of from 40° to 65°); (7) general expressions for perturbations of the second derivatives; (8) local effects, Eötvös formulas; (9-15) Nikiforov's formulas for the calculation of local effects; (16) cartographic effects; (17) Belov's grapho-analytical method of calculating local effects.

Chapter 3, Interpretation of the results of observations. Case of bodies of regular geometric form. Direct and inverse problem of geophysics. (1) The problem; (2) the case of a homogeneous sphere; (3) the case of a homogeneous circular cylinder; (4) the case of a horizontal layer; (5) sensitiveness of the method; (6) the case of an infinitely long horizontal layer of a rectangular shape; (7-8) the case of an inclined layer; (9) a system of parallel, inclined layers of various density; (10) the direct and inverse problem of geophysics.

Chapter 4, The interpretation of the results of observations on the basis of the values of the gradients of the horizontal components of attraction. (1) The case of a homogeneous sphere; (2) the case of a homogeneous circular cylinder lying horizontally; (3) the case of an infinitely long horizontal layer of a rectangular shape; (4) a system of parallel, inclined layers of various density.

Chapter 5, The determination of local effects and perturbances using the Hutton-Hayford's graphical method. (1) The application of the method; (2) the determination of the local and cartographic influences; (3) the determination of the influence of the subterranean masses. -- Ayvazoglou.

A SIMPLIFIED METHOD OF THE CALCULATION OF THE ZERO POINT OF A TORSION BALANCE

By P. Nikiforov

Bull. Inst. Pract. Geophys., No. 2, Leningrad, 1926, pp. 203-231.

A method based on the general formulas of the damping of the oscillating movements of a gravitational variometer is described. Elaboration of observations made on a large scale with the gravitational variometer has proved that V (damping coefficient) depends on the resistance of the air and the elastic property of the torsion filament and remains constant not only for the same series of oscillations in a given azimuth of the beam but also for all series of oscillations in different azimuths. Supposing V constant, simplified equations for V, A_k (successive amplitudes of the curve) and Y_0 (ordinate by which the state of equilibrium is characterized) are derived.

An example for calculations of Y_0 for one series of oscillations based on the measured values of Z_k (ordinates of successive maxima and minima of the curve) is given.

If the value of V for a given instrument changes in the course of time, this can be noticed immediately by the fact that the individual values of Y_0 do not remain constant in the given series of oscillations, but decline consequently from the mean value.

The necessary change of V for obtaining correct results may be determined easily.

For calculations according to this method, tables for the values of A_k, for changes of $\Delta Z = Z_k - Z_{k+1}$ and V are arranged by Prof. A. V. Terentiev and added to this article. -- W. Ayvazoglou.

A METHOD OF RAPID CALCULATION OF THE VECTORS OF THE EÖTVÖS DIFFERENTIAL CURVATURE IN THE CASE OF A SLOPING STRATUM

By K. A. Kirillov

Bull. Inst. Pract. Geophys., No. 2, Leningrad, 1926, pp. 265-269.

Tables and graphs are given from which the values of the vectors of the Eötvös differential curvature in the case of a sloping stratum may be calculated. The article is written with reference to Nikiforov's work, "The Physical Principles of the Gravitational Method of Prospecting." -- W. Ayvazoglou.

THE GREATEST POSSIBLE DEVIATION OF THE INTENSITY OF GRAVITY AND THE DENSITY OF THE CLOSE-MESHED NET OF PENDULUM STATIONS

By Karl Jung

Ztschr. fuer Geophysik, Jahrg. 3, Heft 4, 1927, pp. 137-157.

The fact that the pendulum measurements performed by Berroth above the salt bed in Oldan-Hambühren have proved the possibility of an accurate prospecting of the terrane by means of a close-meshed net of pendulum stations induced the author to give a mathematical solution of this question.

The disposition of the work in the article:

A. The problem. B. Theoretical investigations: (1) Definition and simplified assumptions; (2) mathematical disposition, definition of the variation problem; (3) solution of the variation problem; (4) remarks on the elimination of some simplified assumptions. C. Numerical considerations with regard to their practical application: (1) The greatest possible deviation of the intensity of gravity; (2) the density of the close-meshed net of pendulum stations; (3) justification of the most essential simplified assumptions; (4) examples.

The calculations are illustrated by seven diagrams. -- W. Ayvazoglou.

THE DEPENDENCE OF THE GRAVITATIONAL EFFECT ON THE INTERMEDIATE LAYERS

By T. Schlomka

Ztschr. fuer Geophysik, Jahrg. 3, Heft 8, 1927, pp. 397-400.

The author tries to establish by experiments the existence of the "Refraction of the gravitational lines of forces," as most of the previous theoretical and experimental researches on the dependence of the gravitational effect on the intermediate layers were based on the "absorption of this effect." The arrangement of the experiment (shown in a diagram) and the procedure for it

is explained. For effectuating the gravitation, the author used an iron block of about 1,200 kilograms; the intermediate layer was water (about 825 liters) filling a sheet-iron box of the shape of an isosceles triangle (base ≈ 100 centimeters, opposite angle 37.5°, height 112 centimeters). Measurements were performed by the Eötvös torsion balance and a table of data obtained by a series of experiments is given. Although very accurate absolute values of the effect could not be obtained, the existence of the refraction was confirmed. The author expects to take up the experiments again as soon as all precautions necessary for obtaining accurate data are accomplished. — Ayvazoglou.

ILLUSTRATION OF THE PERIODS OBSERVED, AS WELL AS OF THEIR ACCURACY (With 9 diagrams)

By J. Bartels

Ztschr. fuer Geophysik, Jahrg. 3, Heft 8, 1927, pp. 389-397.

The author proposes for the illustration and comparison of geophysical periods of equal length a simple form of vector diagram by which the amplitude as well as the occurrence of the maximum is shown. Several examples by which the utilization of this "period watch" is shown are discussed in the article and formulas for the estimation of errors in periods observed derived. The graphic representation of the accuracy is made by using the limits of the error obtained by calculations. The contents of the article are arranged as follows: (1) The reason why the use of the period watch shall be preferred to other methods; (2) calculations concerning the arrangement of the period watch; (3) examples given with regard to observations made in different places; (4) determination of errors; (5) relation to periodograms; (6) application illustrated by further examples and diagrams. — W. Ayvazoglou.

A GRADUATED PLATE (PALLET) FOR MEASURING THE ZERO POINT POSITION OF A TORSION BALANCE

By S. Ghirin

Bull. Inst. Pract. Geophys., No. 3, Leningrad, 1927, pp. 317-322.

The determination of the zero-point position in the gravitation variometer with uninterrupted registering of the oscillation of the scale-beam requires computation. The present article offers a method of determination of the required equilibrated position of the scale-beam by means of a special plate (pallet) excluding the necessity of any calculation.

The construction of the graduated plate is carried out according to mathematical calculations obtained from the equation of the oscillations of the scale-beam as they die away. Both branches of the curve are laid out on a glass

plate on which a graduated scale is set. A figure given in the article shows the pallet and the superposed curves of registration given by the apparatus. Checking by means of the graduated plate of the accuracy of reading as compared with other methods has given favorable results. -- W. Ayvazoglou.

2 - MAGNETIC METHODS

ARKANSAS MAGNETOMETER STUDY RESULTS: INSPECTION OF PLATES SHOWS SEVERAL PRONOUNCED HIGHS AND LOWS, FAVORABLE AREAS FOR GAS AND OIL SEARCH

By L. Spraragen

Oil and Gas Jour., vol. 27, No. 36, Jan. 24, 1929, pp. 30, 95; vol. 27, No. 39, Feb. 14, 1929, pp. 42, 110.

The observations of the vertical intensity of magnetism made by the United States Coast and Geodetic survey are taken as the basis of the Arkansas Magnetometer study and are corrected for latitude and longitude. Isogams are drawn, both on the corrected and incorrected values. An attempt is made to show some connection between the magnetic highs and lows of those maps and the oil possibilities of the state as outlined by Branner.

Although the isogams of plate 2 are suggestive as outlining the eastern and southern edge of the Paleozoics, the net of stations is far too sparse for a study of this type to have any appreciable value for the oilman. -- Donald C. Barton.

MAGNETOMETER STUDY OF STATE OF TEXAS, METHOD OF GEOPHYSICAL EXPLORATION BEING USED EXTENSIVELY IN WEST TEXAS, PANHANDLE, AND RED RIVER UPLIFT

By L. Spraragen

The Oil Weekly, vol. 27, No. 31, Dec. 20, 1928, pp. 25, 74-76; vol. 27, No. 33, Jan. 3, 1929, pp. 34, 76, 102.

The scattered net of observations of the vertical magnetic intensity taken by the United States Coast and Geodetic Survey, mostly at county seats, have been used as the basis of the study. The values have been corrected for latitude and longitude and isogams have been drawn both on the corrected and incorrected values. A net of cross profiles has been plotted. Conclusions are expressed in regard to the significance of some of the magnetic anomalies of the map, and the conclusion is stated that certain large features such as faults and ridges - for example, the Mexia fault - can be traced with the magnetometer.

Although a study of this type does reveal certain relations and although it is of value in the hands of an expert geologist - geophysicist - who is aware of its extremely serious limitations, it is extremely and seriously

dangerous. The net of stations is too wide to allow the drawing of such conclusions as have been drawn; as a geologist and geophysicist, the reviewer can not accept many of these conclusions. -- Donald C. Barton.

NEW CALIBRATING DEVICE AIDS WORK WITH THE MAGNETOMETER

By L. Spraragen

Oil and Gas Jour., vol. 27, No. 40, Feb. 21, 1929, p. 118.

A calibrated coil is used to set an electromagnetic field of controlled strength. The device consists of the calibrated coil and an instrument panel with an ammeter, potentiometer, a set of resistances, switch, and 2 dry cell batteries. The coil is so designed to fit around the magnetometer. -- Donald C. Barton.

MAGNETIC MICROLEVELLING CARRIED OUT IN THE IRON ORE DISTRICT OF LIPETZK IN 1925

By E. Krakau, N. Malinine, and M. Penkevicz

Bull. Inst. Pract. Geophys., No. 2, Leningrad, 1926, pp. 83-95.

The authors occupied 237 stations with Brunner-Chasselon magnetic thedolites and Dover dipcircles. The tables annexed at the end of the report contain numerical data concerning the results of observation. -- Donald C. Barton.

THE MAGNETIC OBSERVATORY IN THE COAL BASIN OF THE DON AND THE MAGNETIC SURVEY NEAR MAKEYEVKA

By J. Bahurin

Bull. Inst. Pract. Geophys., No. 2, Leningrad, 1926, pp. 96-106.

The Institute of Practical Geophysics suggests the importance of a magnetic observatory in the Don coal basin for the registration of the variation of magnetic declination in connection with mine surveying and for geophysical prospecting in the southern part of the Union of Socialist Soviet Republics. -- Donald C. Barton.

MAGNETIC OBSERVATIONS IN THE REGION OF THE COAL FIELDS OF THE BASIN OF THE DON (DISTRICT OF THE VILLAGE OF MAKEYEVKA)

By N. Rose

Bull. Inst. Pract. Geophys., No. 2, Leningrad, 1926, pp. 107-123.

Observations were made for the purpose of selecting a suitable site for the magnetic observatory. -- Donald C. Barton.

THE WORK OF THE INSTITUTE OF PRACTICAL GEOPHYSICS ON THE BROWN IRON ORE DEPOSITS (OF THE TULA AND LIPETZK DISTRICTS)

By J. Bahurin

Bull. Inst. Pract. Geophys., No. 2, Leningrad, 1926, pp. 65-82.

Bahurin's paper gives the results of magnetic surveys of iron ore deposits in the Tula and Lipetzk districts, with three maps. Although most of the anomalous vectors only slightly exceed the probable error of observation, a few are considerably larger and seem to correlate with the ore deposits. In one area of the Lipetzk district the anomaly in the inclination amounted to 20 feet and the anomaly of horizontal intensity amounted to 250 gamma. In the Tula district the anomaly in the inclination was 34 feet, in the horizontal intensity 180 gamma, and in the vertical intensity 500 gamma. -- Donald C. Barton.

THE NEW MAGNETIC UNIVERSAL BALANCE

By H. Haalck

Ztschr. fuer Geophysik, Jahrg. 3, Heft 2/3, 1927, pp. 58-68.

A new instrument is described by which the local variations of the three components (declination, horizontal intensity, and vertical intensity) of the earth magnetic field may be determined in a simple and quick way with an accuracy answering that of Schmidt's field balance used for measurements of the vertical intensity.

A detailed description of the instrument, accompanied with a few schematic pictures, and a brief discussion of the theory of the instrument are given. (The detailed discussion of this theory may be found in H. Haalck's work "Ein neues erdmagnetisches Universalvariometer, Zeitschrift fuer Instrumentenkunde, 1927, Heft 1).

The procedure for the observation work: (a) Measurement of the vertical intensity; (b) measurement of the declination; (c) measurement of the horizontal intensity; (d) determination of the constants of the instrument and of the temperature coefficients; (e) observations and calculations (forms of observation and calculation schemes are given).

According to the author one complete measurement, inclusive of the adjustment of the instrument, by an observer more or less familiar with the work, requires from 20 to 25 minutes.

The instrument has proved to be satisfactory for measurements of weak as well as strong earth magnetic variations. (The instrument is manufactured by the Exploration G. M. b. H. Berlin, Linkstrasse 25). -- W. Ayvazoglou.

3 - SEISMIC METHODS

THE SEISMIC METHOD OF MAPPING GEOLOGIC STRUCTURE

By Donald C. Barton

Geophysical Prospecting, 1929. Am. Inst. Min. and Met. Eng., 1929, pp. 572-624.

The velocity of transmission of the elastic earth waves ranges from 2.0$\pm$ kilometers per second in Pho. Pleistocene sediments in the Gulf Coast area to 5.0$\pm$ kilometers per second in the salt. From the point of an artificial explosion at the surface, the wave front travels radially in all directions through the surrounding formations. If a formation with a higher speed of transmission of the waves is present in the subsurface, the elastic waves are (a) reflected directly back to the surface or (b) refracted along the upper surface of the formation and then re-refracted back to the surface. By observation of the time of arrival of the "surface" and reflected waves, the depth to the top of high speed formation may be calculated. By observation of the time of arrival of the "surface" and refracted waves and by construction of time-distance curves, and by proper location of stations, the speed of transmission of the waves in both formations, the depth to the top of the lower (high speed) formation, the depth of the top of that formation, and the lateral limits of that lower formation can be calculated. The mathematical formulas are developed for the case in which each formation is isotropic and homogeneous. The method may be used in reconnaissance for salt domes, in detailing salt domes, and in mapping various types of geologic structure. Crooked drill holes may be mapped by the electric types of seismographs. The seismographs used are mostly of two types; photographically recording mechanical seismographs and photographically recording electric-induction seismographs. The time of the explosion is obtained by wireless, and in reconnaissance for salt domes the distance between shot and receiver is obtained by use of the travel time of the air waves. Charges range from a few to many hundred pounds of 60 per cent dynamite, dependent on the type of work and the local conditions of terrane and wind. The shot lengths range, in general, from 500 feet to 8 miles. In reconnaissance for salt domes, "fan" shooting is used; in most of the other type of work, profile shooting is used. A seismic troop usually consists of one firing unit and three or four receiving units. The cost of seismic work ranges from $10,000 to $20,000 per month. In reconnaissance for salt domes, 150,000 to 300,000 acres per month can be covered. The results of seismic prospecting in the Gulf Coast area have been most brilliant. In comparison to 42 old known domes, 21 domes have been discovered

by the seismograph and proved by drilling and 27 domes have been discovered and not yet tested. Oil fields have been found on six of the 21 domes. In East Texas, 11 new domes were discovered and of those five have been proved by drilling. About 30 seismograph troops are operating in United States and Mexico. -- Author's abstract.

SOUTH LOUISIANA HAS REAL FUTURE IN NEWLY FOUND DOMES

By Jack Logan

The Oil Weekly, vol. 52, No. 7, Feb. 1, 1929, p. 84.

Logan gives a brief review of the results of geophysical prospecting and subsequent drilling in southern Louisiana. Of the 80 wells being drilled, 19 are on "geophysical" domes. Of the "geophysical" domes, 3 have been proved commercially productive, 2 have a little oil, 6 have been drilled without finding oil as yet, and 19 are untested. -- Donald C. Barton.

HOME OF "GATOR" AND WATER LILY OPENED FOR OIL BY SCIENCE, "DYNAMITE BUGGY" AND PIROGUE ARE TOOLS WITH WHICH WILDCATTERS SEEK DOMES IN LOUISIANA SWAMPS

By Wallace Davis

The Oil Weekly, vol. 52, No. 27, Feb. 1, 1929, pp. 69-76.

A popular account of field practice with the seismic method in the Louisiana swamps, illustrated by many interesting photographs of characteristic phases of the work. -- Donald C. Barton.

THE SEISMOGRAPH IN THE GULF COAST

By Mark C. Malamphy

The Oil Weekly, vol. 52, No. 5, Jan. 18, 1929, pp. 31-34.

An excellent succinct popular statement of the seismic method of prospecting for salt domes as it is actually used in the Gulf Coast. The paper is illustrated with a characteristic "shot" map and with time-distance curves. -- Donald C. Barton.

SEISMIC EXPERIMENTS WITH EXPLOSIONS; PRELIMINARY NOTE

By P. M. Nikiforov

C.R.Acad.Sci.de l'U.S.S.R.,Ser.A, Leningrad, Oct., 1926, pp. 189-190.

The author briefly reports experiments carried on with various seismographs employing (1) a vertical seismograph of B. B. Galitzin, slightly changed for this special work: (2) the horizontal seismograph by Wiechert-Mintrop: (3) a small horizontal seismograph of their own design. Problems studied are defined, results obtained will be given in a separate paper at an early date. -- E. U. Von Buelow.

4 - ELECTRICAL METHODS

BASIS FOR CALCULATING THE OBSERVATIONS OF EARTH CURRENTS

By A. Petrowsky

Bull. Inst. Pract. Geophys., No. 2, Leningrad, 1926, pp. 124-142.

A continuation of Petrowsky's previous paper "The Theory of Earth's Current Measurement," in which the mathematical expression was deduced for the potential in the zone surrounding a spherical electrode half sunk in the ground, and gives an analysis of that formula and a method for its application to the distribution of the field within the medium according to the deviation of a galvanometer included in the circuit between a pair of such electrodes. The observations in the fields at Ridder's mine("Electrometric Methods of Ore Prospecting and Experimental Investigation at Ridder's Mine during Summer of 1924." The abstracts of these two articles are given in Geophysical Abstracts No. 1.) confirmed the theoretical deductions. -- Donald C. Barton.

ELECTRICAL METHODS OF PROSPECTING

By J. B. Mawdsley

British Columbia Miner, vol. 2, 1929, pp. 21-27.

The general principles underlying the self-potential, equipotential and electromagnetic methods are discussed. The cost of electrical surveying and the recent work of the Canadian Geological Survey are briefly dealt with. The paper is of a general type. -- D. A. Keys.

ELECTRICAL PROSPECTING BY MEANS OF ALTERNATING CURRENT
(With 8 Diagrams)

By Richard Ambronn

Gerlands Beirträge zur Geophysik, Band 19, Heft 1, 1928, pp. 5-58.

In investigations of the distribution of electric conductivity in the subsoil by means of alternating current, more particularly for the purpose of the geophysical location of minerals, salt-water horizons, and so forth, due consideration has not hitherto been given to the elliptic polarisation of the voltage, current, and magnetic field vectors, which was first recognized and used in field work by the author. The electromagnetic vectors of such a distribution of alternate currents in space describe plane ellipses of the frequency of the exciting field, the quantitative measurement and representation of which alone reproduce a complete and unambiguous picture of the whole phenomenon.

Such a complete measurement and representation of the electromagnetic field of the earth currents is possible by means of several systems of observational data. One may utilize either the plan of the ellipse, the magnitude and direction of the axes and the phase of one diameter, or one may define the plane of the vibration ellipse by means of the amplitudes and phases of three linear vibration-components not lying in a common plane.

Various combined methods are also suitable, both for practical determinations in the field and also for a clearly understandable, graphic representation of the results. The formulas for the calculation of various systems of this kind are derived in the article.

The vibration ellipse completely represents the phenomenon in time and space at any given point of observation, whilst the momentary fields, calculated for a definite and suitably chosen time - phase - give a clear view of the structure of the field at any such time.

By vectorially subtracting from the field so measured, the normal field which would have been obtained above a homogeneous subsoil, one obtains a derived, elliptically polarized field of disturbance in which the influence of the required electromagnetic anomalies in the subsoil is sharply defined. For this field of disturbance the momentary fields at suitably chosen times can also be calculated and diagrammatically represented, so that they may be individually analyzed.

On comparing this perfected process with those methods of electric investigation of the subsoil which have hitherto been used, it will be seen that heretofore only very crude determinations have been obtained, which are, in fact, usually quite erroneous physically, as the directions characterized as "lines of force" are not even lying in the plane of the true vibration ellipses. It is only by taking into account the elliptic polarization of the electromagnetic field of the earth currents that it is possible to carry out a scientifically accurate investigation of the subsoil by alternating current. -- Author's Abstract.

WHEN AND WHERE SHALL THE ELECTRICAL METHOD OF PROSPECTING BE USED?

By Ingemar Tennberg

Berg und Hüttenmännisches Jahrbuch, Band 76, Heft 2, Wien, 1928.

The author discusses the question why so many persons interested in exploration of ore deposits are not enthusiastic about the electrical method of prospecting, notwithstanding the fact that ore deposits have already been found by using this method. In his opinion, this distrust may be explained in the first place by the fact that the prospecting is often performed by persons and firms who do not know much about geophysics. But even in case of prospecting by substantial firms, the mine owners are often not satisfied with the results, as they expect too much.

The work of a geophysicist can not be compared with that of a bridge constructor or a surveyor of mines, as the latter two work with known qualities, whereas the solution of the problem assigned to a geophysicist depends mostly on his theoretical knowledge of the question and his personal ability.

In most cases the possibility of the solution can not be determined in advance.

The first supposition for a successful electric prospecting is that of a sufficient difference in conductivity between the ore body sought for and the bodies surrounding it. How great the ratio of conductivity shall be is at present a question discussed by the two adepts of the geophysic science, Ambronn and Koenigsberger.

The following ores are considered to be the most suitable for electric prospecting: Galena, chalcopyrites, bornite, sulphur-pyrites, magnetic pyrites, arsenic pyrites, arsenical pyrites, gray copper ore, some heavy metal sulphates, and all precious and semiprecious metals. The specific resistance of pure metals is between 100 ohms and a fraction of one ohm per cubic cm. The resistance of ore bodies lying close to the surface of the earth seldom has values smaller than 3,000 ohms per cubic centimeter. Thus a ratio of the conductivity of the rocks and that of the ore of from 1/30 upward may be adapted. A certain minimum dimension and maximum depth should also be established, as the ratio of the conductivity depends on them. According to the opinion of the author, an ore body with a ratio of at least 1/1,000, lying at a depth half as great as its length, can be determined from the surface of the earth with certainty. The minimum length of the ore body should be, according to experience, from 15 to 20 meters. In case of larger ore bodies or very small depths, the ore may be found if the ratio of conductivity is even smaller. But as the ores are mostly found disseminated in dead rocks, the determination of the ratio of conductivity is difficult, thus the possibility of solving the problem can not be decided in advance. In addition to the cardinal points such as: Ratio of conductivity, dimension, depth, and the influence of the distribution of the parts of the ore

body, a certain importance for electrical prospecting must also be assigned to the absolute values of the conductivity of the ore bodies and rocks. In case of specific resistance of rocks smaller than from 4,000 to 5,000 ohms per cubic centimeter, complications in using the ordinary alternating current process may occur which can be overcome only with the aid of the most modern technical means, available only to the firms equipped the best. In conclusion, the author recommends the use of the geo-electrical method of prospecting, by all means, in places in which large sulphide compact ore bodies (except the pure zinc-blende) are deposited in eruptive rocks or crystalline schists. In case of sedimentary rocks exhibiting only a slight metamorphism or not metamorphic at all, the task is more difficult but may be accomplished by organizations equipped with the best scientific means. Gangue deposits must be considered unfavorable objects. Single small lenses can be discovered only if they are connected one with another so that a continuous electric conductor is formed. Only occasional results may be obtained with impregnated ores, the parts of which are bedded inside of isolating masses. As far as the preference, in connection with sensitiveness of one or the other method is concerned, the author's opinion is that a method by which the material observed may be utilized in the most logical and mathematical way should be chosen. -- W. Ayvazoglou.

5 - RADIOACTIVE METHODS

ON HIGHLY PENETRATING EARTH RAYS

By L. Bogoyavlensky

Bull. Inst. Pract. Geophys., No. 2, Leningrad, 1926, pp. 184-195.

Ninety stations in the Caucasus were occupied over an area of 10,500 square meters of radium deposits in fissures in travertine. The curves of equal intensity were elongated in the same direction as the fissures. The intensity measured seemed to be constant within 3 per cent, no matter whether the observations were taken a year apart or under different meteorological conditions. At 22 of the stations the radiations were absorbed by lead screens 1 centimeter thick. The presence of radiation of greater penetrating power than the gamma rays of radium C were indicated. The effect of the radioactivity of the air was nil and of the upper soil negligible. -- Donald C. Barton.

MEASUREMENTS OF RADIOACTIVITY AS A GEOPHYSICAL METHOD OF PROSPECTING

By Ferdinand Müller

Ztschr. fuer Geophysik, Jahrg. 3, Heft 7, 1927, pp. 330-336.

The enrichment of radioactive substances occurs under certain geological conditions. In Müller's article the steps necessary for the determination of these enrichments are described and some examples of the application given. The following problems are examined:

A. Preparatory facts. The author mentions the literature concerning the researches and distribution of radioactive substances in minerals, rocks, waters of all kind, atmosphere, and soil. According to this distribution, there are two main kinds of problems of radioactive measurements with regard to geophysical prospecting; the first consists of discovering the active minerals and sources and of the determination of the amount and kind of the acting constituent; the second, of the solution of special geological questions in connection with the radioactive enrichment.

B. Methods of investigation: (1) Ambronn's method; (2) Bogoyavlensky's method; (3) Koenigsberger's improvement on Ambronn's method; (4) method by which the emanation content of the soil gases is determined by introduction of them into the ionization chamber by means of a pump; (5) measurements of emanation by means of fontactoscopes or fontactometers; (6) method of comparison of test pieces taken from place to place.

C. Practical examples concerning the use of one or the other method with diagrams of places in which investigations were made are given in the article. The author adds besides some observations of his own during his practical research work on behalf of the Elbof-Piepmeyer Co. -- W. Ayvazoglou.

RADIOMETRIC EXPLORATION OF OIL DEPOSITS

By L. N. Bogoyavlensky

Bull. Inst. Pract. Geophys., No. 3, Leningrad, 1927, pp. 113-124.

Based on the results of the study of emanations of the earth, the author performed experiments in the oil deposits of the Maikop district (Province of Kuban) in order to verify his supposition that an extremely slight change of radium content in ores influences the intensity of penetrating radiation and that the radioactivity of an oil bed must differ from that of the strata surrounding it, on the following grounds: (1) Oil, being an organic compound, possesses an immense power of absorption of radioactive emanations; and (2) the strata underlying the oil bed, having been developed from repositories of sea-ooze, are richer in radium because of the greater power of absorption peculiar to colloids. The data obtained from the experiments, illustrated by the diagrams of sections and intensities, proving the correctness of his supposition, are given in the article. -- W. Ayvazoglou.

MEASUREMENT OF THE CONTENT OF RADIUM EMANATIONS IN THE ATMOSPHERIC AIR

By A. Lomakin

Bull. Inst. Pract. Geophys., No. 3, Leningrad, 1927, pp. 124-136.

The author describes the measurement of radium emanations in the atmospheric air, using the aspiration method, by means of blowing the tested air through a cylindrical condenser. A scheme of measurements based on theoretical

calculations and a diagram is given. A special type of field installation arranged for this purpose is described; the construction of the apparatus is given schematically. In order to obtain the necessary intensity an automobile magneto has been used. The consumption of the air was increased up to 70 liters per second (only from 3 to 8 liters per second in previously known field installations), thus the time required for a full measurement was reduced from the usual two to three hours to 30 to 40 minutes. The records of values obtained by measurements in Leningrad and in Piatigorsk (Caucasus) are given in a table. -- W. Ayvazoglou.

EXPERIMENTS ON RADIATIONS PENETRATING THROUGH THE CRUST OF THE EARTH

By L. N. Bogoyavlensky and A. A. Lomakin

Bull. Inst. Pract. Geophys., No. 3, Leningrad, 1927, pp. 87-111.

Disposition of the work: (1) Apparatus used; (2) place of experiments; (3) constancy of the tension of the penetrating radiation (tables showing data of observations made during three continuous years); (4) influence of the radioactivity of the atmosphere; (5) influence of cosmic radiation coming from above; (6) influence of the radioactivity of the upper stratum of the soil (diagrams and tables with data of observations at different points using hoods and filters of different thickness); (7) calculation of the coefficient of absorption with formulas and tables showing this absorption for hoods and filters; (8) calculation of the length of the wave.

The experiments were performed in Piatigorsk (Caucasus) by means of a portable electometer covered with lead 1 centimeter thick. The measurement proved that by using the same apparatus the intensity varied greatly, depending on the stations of observation. These variations of intensity were especially marked in places rich in radium, where divergences of about 100 per cent were sometimes observed between stations separated one from another by a few meters only. Measurements made at the same stations of observation during a period of three years have shown that the intensity was constant and independent of meteorological conditions. The content of radio elements in the upper layer of the soil was proved by the measurements to be constant, thus the variations were due to deeper strata only. Four hoods, fitting one into another, 2 centimeters thick each, were used for covering the apparatus from above and from the sides, and four flat lead screens (filters) were used for protecting the apparatus from beneath; the experiments have proved that the influence of the hoods was very slight only and that of the filters important, thus testifying that the electrometer was influenced chiefly from below. The coefficient of absorption was calculated by varying the thickness of the lead protecting the apparatus, from 0 centimeter to 8 centimeters. The coefficients of absorption by filters varied from 0.45 to 0.06 for 1 centimeter. In most cases the coefficient of absorption decreased with the increase of the thickness of the lead. -- W. Ayvazoglou.

6 - GEOTHERMAL METHODS

By J. Joly

Gerlands Beiträge zur Geophysik, Band 19, Heft 4, Leipzig, 1928, pp. 415-441.

After a short historical introduction the following matters are dealt with: The nature and structure of the continents; the distribution of radioactivity in the earth's surface structure; the thickness of the continents as estimated on isostatic grounds; the distribution of temperature in the continents; seismic evidence as to the earth's surface structure; the probable origin of the surface structure; the thermodynamics of a revolution are specially dwelt on and a paper "On the Thermal Instability of the Earth's Crust" by H. H. Poole and J. H. J. Poole is quoted, in which the recent accessions to our knowledge of the physical properties of rocks are applied to the views of Lord Kelvin and the conclusion is reached that cyclical discharge of radioactive heat must occur.

There follows a reference to the intervention of tidal movements in the discharge of collected heat. An abstract of a mathematical paper by Cotter, "On the Escape of Heat from the Earth's Crust," is succeeded by an estimate of the rate of thermal genesis in the substratum. The stability of the ocean floor is referred to, and finally there is a note on the experimental reproduction of cyclical thermal system. -- Author's Abstract.

7 - UNCLASSIFIED METHODS

AFTER DEEP DOMES WHAT?

By John F. Weinzierl

The Oil Weekly, vol. 52, No. 8, Feb. 8, 1929, p. 34.

The change in emphasis on the seismic method back to the torsion balance in the Gulf Coast area is mentioned. The opinion is expressed that a framework of detailed pendulum observations is necessary to explain the deeper tectonics. -- Donald C. Barton.

METHODS OF APPLIED GEOPHYSICS FOR THE EXPLORATION OF OIL, ORES, AND OTHER USEFUL DEPOSITS

By Erich Pautsch

Published privately by the author, Houston, Tex., 1927, 82 pages; numerous diagrams.

In the few pages of this small book Pautsch covers: (1) The methods and tasks of applied geophysics; (2) the gravitational method, giving a brief discussion of the elements of the theory of gravitation and a more extended discussion of the theory of the Eötvös torsion balance method with some of the more

basic formulas and profiles of the gradient and differential curvature produced by different types of bodies; (3) the seismic and acoustic methods, a discussion principally of the theoretical principles of (pure rather than applied) seimology and with an inadequate description of the instruments actually in use; (4) the magnetic method, theory and instruments, a very brief description and discussion; (5) the electric methods, a brief history of the use of electric methods and a discussion of the galvanometric method, the inductive method, the theory and description of the current in the earth method, resistance of some conducting bodies method, the (wireless) wave method, the capacity method, and the polarization method.

The discussions are so condensed and brief that the beginner will doubtfully follow and understand the discussion, and the advanced student of geophysics will not find it of much use. -- Donald C. Barton.

ELEMENTS OF GEOPHYSICS, AS APPLIED TO EXPLORATIONS FOR MINERALS, OIL AND GAS

By Richard Ambronn (M. C. Cobb, translator)

372 pp., 84 figs., 1,672 refs. Translated from "Methoden der Angewandten Geophysik," Liesegangs Wissenschaftliche Forschungsberichte, Naturwissenschaftliche Reiche, Band 15, Theodor Steinkopf, Dresden and Leipzig, 1926. McGraw Hill Book Co., New York, 1928.

The purpose of the book is not to serve as a textbook for the execution of geophysical surveys but is, first, to provide a complete and well-balanced review of all the geophysical methods that may be used in economic geology, so that the geologist, mining engineer, layman, or executive who has to do with them or is considering the use of them, may come as completely as possible to understand their possibilities and applicabilities; and, second, to provide for the practical geophysicists and others interested in applied geophysics, reference to the thought and literature on the subject.

The book comprises an introductory chapter on the development and present position of geophysical methods in prospecting; influence of the subsurface formations on the character of the gravitational field at the surface of the earth, including pendulum and torsion-balance surveys; magnetic methods of investigations; the use of radioactive and atmospheric-electric measurements for geophysical prospecting; electrical methods of prospecting in the earth's interior; and the use of temperature methods in applied geophysics.

Under each subject the elements of the pure as well as the applied theory are given, the geophysical instruments and their method of use are described, the results of many applied geophysical surveys are shown, and practically all published descriptions of geophysical surveys are mentioned. The literature on geophysics apparently has been thoroughly searched and a detailed review is given of the pertinent thoughts, theories, suggestions, observations, data, and at the point of mention, direct references are made to the literature. There are 84 diagrams, sections, and maps and a bibliography of 1,672 articles from 277 serials.

The portions of the manuscript checked against the original by the reviewer were found faithfully to follow the German text. Minor revision and small additions of new material were made in a few places in the book by Ambronn during the translation, but in general the book is as of the date of the original, March, 1926, and the extensive literature of 1926, 1927, and first half of 1928 is not mentioned or listed. The typography is good and the illustrations clear.

The only adverse criticism which the reviewer has to offer is that in some places not sufficient distinction is made between practically untried methods which theoretically should work and methods which have actually been proved successful. The general impression which the book gives in regard to the successful applicability of the methods in economic geology is slightly too optimistic. The book also would have been more convenient in many ways if it had been pocket-sized like the German original.

If the reviewer were allowed only one book on geophysics he would have to choose this. It will be the indispensable reference book of the professional geophysicist and the textbook of the elementary and advanced students of geophysics and of geologists, executives, and others interested in applied geophysics. -- Donald C. Barton.

APPLIED GEOPHYSICAL METHODS IN AMERICA

By Donald C. Barton

Economic Geology, vol. 22, No. 7, November, 1927, pp. 609 to 668.

The methods of applied geophysics in practical use in economic geology in America are largely foreign-born. The redeeming feature to an American is the way that they have been taken up by the American oil companies. The methods in considerable practical use are four - the Eötvös gravimetric, the seismic, the electric, and the magnetic. The Eötvös gravimetric method was developed by Baron Eötvös of Hungary during the latter part of the past century. The seismic method was worked out by Fessenden of the United States in 1913, but in its present development is due largely to a later but independent development of the method by Mintrop of Germany. The application of electric methods to prospecting has been the subject of experimentation for nearly a century; the recent successful developments of practical instruments and technique for commercial application of the methods was by Lundberg and Nathorst in Sweden, Schlumberger in France, Gella in Austria, and Sundberg in Sweden. The magnetic method has been in practical use for over half a century in connection with magnetic iron ores. The present application of the method to faintly magnetic ores and to geologic structure dates from the recent invention by Schmidt of Berlin of his adaptation of the Lloyd balance (British) and from Schuh's mapping of the Lübteen salt dome in Germany with a Schmidt-Lloyd balance. These methods have been taken up rapidly by the American oil companies. But the recognition of the methods by schools and institutions has been much greater in Europe than in America. The record of these methods in prospecting in the Gulf Coast area has been brilliant for the torsion-balance and seismic methods. The methods, furthermore, show

interesting potentialities for research in pure science problems of geology. The present geologic status of the methods is that the instruments and technique of observation have been developed much further than the general knowledge of the geologic interpretation of the results. Research on and with the methods is being done almost solely by commercial companies, by whom the results of the research mostly will not be released. These methods offer an interesting and almost untouched field of research to schools both of pure and applied science.-- Author's Abstract.

METHODEN DER ANGEWANDTEN GEOPHYSIK (METHODS OF APPLIED GEOPHYSICS)

By Richard Ambronn

Liesegangs Wissenschaftliche Forschungsberichte, Naturwissenschaftliche Reihe, Band 15, Theodor Steinkopf, Dresden and Leipzig, 1926, 258 p., 1,671 refs.

The purpose of the book is not to serve as a textbook for the execution of geophysical surveys, but to provide a complete and well-balanced, well-evaluated review of all the geophysical methods that may be used in economic geology so that the geologist, mining engineer, layman, or executive who has to do with them or is considering the use of them may come completely as possible to understand them, their possibilities, and their applicabilities, and to provide for the practical geophysicists and others interested in applied geophysics, the reference to all the more important literature of the subject.

The content of the book comprises an introduction giving a brief statement of the history of the development, and of the present status of the geophysical methods of exploration, and chapters on the gravimetric, magnetic, radioactivity, electric, seismic, and geothermal methods.

This book is now available in an English translation "Elements of Geophysics as Applied to Exploration for Minerals, Oil, and Gas." -- Donald C. Barton.

8 - GEOLOGY

THE EFFECT OF GEOPHYSICAL METHODS ON DRILLING IN THE GULF COAST

By Donald C. Barton

The Oil Weekly, Houston, Tex., September 2, 1927.

The use of geophysics in connection with drilling in the Gulf Coast is another example of the replacement of haphazard methods of guess, luck, and repeated trial by moderately precise scientific methods, and marks another step in the evolution of oil production into an engineering science. The use of the geophysical methods in the Gulf Coast has shown that the torsion-balance and

the seismograph methods afford considerable, and in some places, high precision in the location of wells in reference to the salt domes. Although as yet they have not proved practicable in the Gulf Coast, the electric methods of prospecting afford the bare possibility that ultimately they may afford considerable precision in the location of the wells in reference to the oil sands directly. The modern geophysical methods have nothingin common with the ancient but not honorable family of doodlebugs and wigglesticks which have never shown any sign of any ability to be of service to the oil operator. -- Author's Abstract.

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INDEX OF AUTHORS AND SUBJECTS

Circular 6134,
May, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SAFEGUARDING ELECTRICAL EQUIPMENT USED IN GASSY MINES[1]

EUROPEAN PRACTICE: I - GREAT BRITAIN[2]

By L. C. Ilsley[3]

INTRODUCTION

Cooperation between the United States Bureau of Mines and the Safety in Mines Research Board of Great Britain, continuous since 1924, has made possible this and other papers on safety subjects. Grateful acknowledgment is made to representatives of the Safety in Mines Research Board of the Mines Department for their assistance in arranging visits to several mine-safety stations, and to F. H. Wynne, Deputy Chief Inspector of Mines, Great Britain, for arranging visits to mines in Great Britain, Belgium, France, and Germany.

During the summer of 1927 the writer had the privilege of visiting the mine-safety testing stations in Great Britain, Belgium, Germany, and France, in the order named. All of these countries have large coal mines, many of which are rated as gassy. Therefore, when the installation of electrical equipment is contemplated, each of these countries is confronted with the same safety problem as in the United States - the development of electrical equipment that will not ignite gassy mine atmospheres, should such atmospheres through neglect or accident surround the equipment. The means used by those countries in safeguarding gassy mines should therefore be of general interest to safety engineers in American coal mines, and it is proposed to give a brief survey for each of the four countries mentioned. The first of these surveys will cover conditions in Great Britain.

REGULATIONS BEARING ON SAFETY

A good beginning may be made by mentioning some of the requirements found in coal mines act of 1911. This act includes a section, "General Regulations As to the Installation and Use of Electricity," from which the following pertinent paragraphs have been abstracted:

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1 The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be used: "This paper represents work done under a cooperative agreement between the U. S. Bureau of Mines and the Safety in Mines Research Board of Great Britain. Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"
2 This paper is a revision of, and supercedes, Information Circular 6082 bearing the same title and issued in September, 1928.
3 Electrical engineer, U. S. Bureau of Mines.

(119) Notices shall be sent to the Inspector of the Division, on the forms prescribed by the Board of Trade, as follows, namely:

(119.)--(i.) Notice of the intention to introduce apparatus into any mine, or into any ventilating district in any mine.

If the Inspector of the Division does not object in writing, within one calendar month from the receipt by him of the notice, to the carrying out of either of the intentions specified in the first or second notices, the owner shall be entitled to carry out such intention or intentions.

Provided that this regulation shall not apply to telephones and signalling apparatus.

(119.)--(ii.) Notice of the intention to introduce or reintroduce electricity into any mine where the use of electricity has previously been prohibited by Section 60 (1) of the Act.

If the Inspector of the Division does not object in writing, within one calendar month from the receipt by him of the notice, to the carrying out of either of the intentions specified in the first or second notices, the owner shall be entitled to carry out such intention or intentions.

Provided that this regulation shall not apply to telephones and signalling apparatus.

(119.)--(iii.) On or before the 21st day of January in every year an annual return giving the size and type of apparatus and any particulars which may be required by the Board of Trade as to the circumstances of its use.

(125.)--(a.) All metallic sheaths, coverings, handles, joint-boxes, switchgear frames, instruments covers, switch and fuse covers and boxes, and all lampholders, unless efficiently protected by an earthed or insulating covering made of fire-resisting material, and the frames and bedplates of generators, transformers, and motors (including portable motors), shall be earthed by connection to an earthing system at the surface of the mine.

This rule shall not apply (except in the case of portable apparatus) to any system in which the pressure does not exceed low-pressure direct current or 125 volts alternating current.

(125.)--(b.) Where the cables are provided with a metallic covering constructed and installed in accordance with Regulation 129 (e) such metallic covering may be used as a means of connection to the earthing system. All the conductors of an earthing system shall have a conductivity at all parts and at all joints at least equal to 50 per cent of that of the largest conductor used solely to supply the apparatus a part of which it is desired to earth. Provided that no conductor of an earthing system shall have a cross-sectional area of less than 0.022 of a square inch.

(125.)--(c.) All joints in earth conductors and all joints to the metallic covering of the cables shall be properly soldered or otherwise efficiently made, and every earth conductor shall be soldered into a lug for each of its terminal connections. No switch, fuse, or circuit breaker shall be placed in any earth conductor.

(131.)--(a.) Every person appointed to work, supervise, examine, or adjust any apparatus shall be competent for the work that he is set to do. No person except an electrician or a competent person acting under his supervision shall undertake any work where technical knowledge or experience is required in order adequately to avoid danger.

(131.)--(b.) An electrician shall be appointed in writing by the manager to supervise the apparatus. If necessary for the proper fulfillment of the duties detailed in the succeeding paragraphs of this rule, the manager shall also appoint in writing an assistant or assistants to the electrician.

(131.)--(c.) The electrician shall be in daily attendance at the mine. He shall be responsible for the fulfillment of the following duties, which shall be carried out by him or by an assistant or assistants duly appointed under paragraph (b): (i.) the thorough examination of all apparatus (including the testing of earth conductors and metallic coverings for continuity) as often as may be necessary to prevent danger; and (ii.) the examination and testing of all new apparatus, and of all apparatus reerected in a new position in the mine before it is put into service in the new position.

Provided that in the absence of the electrician for more than one day the manager shall appoint in writing an efficient substitute.

(131.)--(d.) The electrician shall keep at the mine a log book made up of daily log sheets kept in the form prescribed by the Secretary of State. The said log book shall be produced at any time to an inspector of mines on his request.

(131.)--(e.) Should there be a fault in any circuit the part affected shall be made dead without delay, and shall remain so until the fault has been remedied.

(131.)--(f.) All apparatus shall be kept clear of obstruction and free from dust, dirt, and moisture, as may be necessary to prevent danger.

Inflammable or explosive material shall not be stored in any room, compartment, or box containing apparatus, or in the vicinity of apparatus.

[illegible]

(b)(4)(b) [illegible] recommended [illegible]

(b)(4)(c) An alternative [illegible] would be [illegible] the recommended [illegible] preceding paragraph [illegible] conditions.

(b)(4)(d) The recommendations ought to be fairly broad [illegible] responsibility for the fulfillment of the following policy [illegible] should be maintained by [illegible] conditions [illegible] the recommended [illegible] procedures [illegible] their [illegible] modification.

[illegible] the [illegible] committee [illegible] situation [illegible]

(b)(4)(e) [illegible] the recommendations could be [illegible] in a [illegible] be [illegible] of [illegible] procedures [illegible]

(b)(4)(f) [illegible] of [illegible] general [illegible]

[illegible] this [illegible] responsibility [illegible]

(131.)--(g.) Adequate precautions shall be taken by earthing or other suitable means to discharge electrically any conductor or apparatus, or any adjacent apparatus if there is danger therefrom, before it is handled, and to prevent any conductor or apparatus from being accidentally or inadvertently electrically charged when persons are working thereon. While lamps are being changed the pressure shall be cut off.

Provided that this paragraph shall not apply to the cleaning of commutators and slip rings working at low or medium pressures.

(131.)--(h.) The person authorized to work an electrically-driven coal-cutter or other portable machine shall not leave the machine while it is working, and shall, before leaving the working place, ensure that the pressure is cut off from the flexible trailing cable which supplies such coal-cutter or other portable machine. Trailing cables shall not be dragged along by the machine when working.

(131.)--(i.) Every flexible cable shall be examined periodically (if used with a portable machine, at least once in each shift by the person authorized to work the machine), and if found damaged or defective it shall forthwith be replaced by a spare cable in good and substantial repair. Such damaged or defective cable shall not be further used underground until after it has been sent to the surface and there properly repaired.

(132.) In any part of a mine in which inflammable gas, although not normally present, is likely to occur in quantity sufficient to be indicative of danger, the following additional requirements shall be observed:-

(132.)--(i.) All cables, apparatus, signalling wires, and signalling instruments, shall be constructed, installed, protected, worked, and maintained, so that in the normal working thereof there shall be no risk of open sparking.

(132.)--(ii.) All motors shall be constructed so that when any part is live all rubbing contacts (such as commutators and slip-rings) are so arranged or enclosed as to prevent open sparking.

(132.)--(iii.) The pressure shall be switched off apparatus forthwith if open sparking occurs, and during the whole time that examination or adjustment disclosing parts liable to open sparking is being made. The pressure shall not be switched on again until the apparatus has been examined by the electrician or one of his duly appointed assistants, and the defect (if any) has been remedied or the adjustment made.

Some of the outstanding features of the rules just quoted that have a chief bearing in maintaining electrical safety are:

1. Strict rules with respect to giving notice to the Mines Department of intention of installing new equipment.

2. Strict rules regarding the earthing (grounding) of all equipment and wiring.

3. Instructions for selection of the electrician and other workmen with the requirement that the electrician must be in constant attendance.

4. Requirement that a daily log must be kept of happenings on electrical apparatus in a prescribed log book.

5. Requirement that defective equipment be put out of service at once.

6. Special rules covering the use of electrical equipment and wiring in gassy portions of the mine to prevent "open sparking."

TESTS AND REQUIREMENTS FOR ELECTRIC MOTORS

The users and manufacturers of electrical equipment are taking forward steps to formulate testing procedure for equipment and to meet the expense entailed in having types of equipment tested.

The following regulation approved and issued in 1926 under the auspices of the British Engineering Standards Association (composed of the Institution of Civil Engineers, Institution of Mechanical Engineers, Institution of Naval Architects, Iron and Steel Institute, and Institution of Electrical Engineers) shows the general trend of testing electrical equipment for safety in Great Britain:

British Standard Specifications for Flame-Proof Enclosures and for Testing Such Enclosures

I. Definition of Flame-Proof Enclosure (Including Explosion-Proof) for Electrical Apparatus.

1. A flame-proof enclosure (including explosion-proof) for electrical apparatus is one which will withstand, without injury, any explosion that may occur in practice within it under the conditions of operation within the rating of the apparatus enclosed by it (and recognized overloads, if any, associated therewith), and will prevent the transmission of flame such as will ignite any inflammable mixture which may be present in the surrounding atmosphere.

Note1.--In the absence of any statement to the contrary, it is assumed that the flame-proof enclosure has to meet the ordinary requirements of the Coal Mining Industry in which the inflammable mixture to be considered will ordinarily contain methane, but in other industries other inflammable mixtures will be encountered. Similarly it will be necessary to consider other inflammable gases in relation to particular apparatus, such as the gas resulting from decomposed oil in oil-immersed switchgear and hydrogen in storage batteries.

Note 2.--In view of the danger which would result from a destructive short-circuit within the enclosure, special attention to details of design and manufacture is necessary. In addition, the protection of the circuit supplying the apparatus should be such as to ensure, as far as practicable, that the highest recognized overload for the apparatus shall not be exceeded,having regard to the amount of destructive energy available at the apparatus calculated from the size of generating plant and the impedance of the circuit between it and the apparatus.

II. Specification of Tests for Various Classes of Apparatus to Prove Compliance with the Definition of Flame-Proof Enclosure.

General

2. In conformity with Note 1 to the Definition of Flame-Proof Enclosure, it is assumed that the apparatus has to meed the ordinary requirements of the Coal Mining Industry in which the inflammable mixture to be considered will ordinarily contain methane.

For some purposes a certificate may be required as to the flame-proof qualities of a casing with respect to such inflammable mixtures as, for example, hydrogen and air. Such a certificate shall be granted only on the results of tests carried out with the particular inflammable gas specified therein.

The design and construction of flame-proof apparatus submitted for test and certificate should comply, as regards flame-proof enclosure, with the B.E.S.A. Specification, if any exists, for such apparatus, but if there is not such Specification then the principles of design and construction, as regards flame-proof enclosure, in any appropriate B.E.S.A. Specification should be observed.

Tests

3. The following tests shall be carried out with the apparatus correctly assembled, with all its parts (including oil, filling compounds, etc., if any) in place, and with all electrical connections from the interior to the exterior of the casing made. Apparatus designed for the protection of rapidly revolving parts, such as a motor casing, shall be tested with such parts running at their maximum working speed.

To meet the ordinary requirements of the coal mining industry, the casing shall be filled at the air temperature and pressure prevailing at the testing station with the most explosive mixture of methane and air (i.e., containing between 9.5 and 10.5 per cent of methane by volume) and shall be surrounded by the most readily ignited mixture of methane and air (i.e., containing between 8.5 and 10.5 per cent'of

methane by volume) at the air temperature and pressure prevailing at the testing station. The tests of flame-proof enclosures in any other inflammable atmosphere than that containing fire damp shall be made with the most explosive mixture of the particular inflammable gas and air at the temperature and pressure prevailing at the testing station. The explosive mixture within the casing shall be ignited, if possible, by the spark produced when an electric current of sufficient intensity is established or broken by the normal mechanical operation of the apparatus. Otherwise, any suitable means may be employed provided that the position of the point of ignition would be produced in the normal working of the apparatus.

It is recognized that until such time as experimental work on the testing of switchgear under short circuit conditions has been completed, it would be unwise to specify that the enclosure should be tested for its flame-proof properties with the apparatus operating under the most severe conditions likely to be met in normal service. The tests specified, therefore, are liable to revision and modification after further research if it is found that testing under short-circuit conditions is essential.

Note:--It is not considered desirable to define the number of tests or the character of any additional tests which the testing officer might desire to make. It is hoped that further instructions can be given after experience has been gained.

Test Certificate

4. An apparatus that satisfies the requirements of this Specification and has passed the tests to which it has been submitted can be considered as complying with the British Standard definition of Flame-Proof Enclosure and a certificate in the form given in Clause 5 should be granted.

Form of Test Certificate

5. The following form of test certificate should be used:--

Certificate of Test as to Flame-proof Enclosure. This is to certify that a .... (description of article), identical in all essential respects as to design, workmanship and material with that indicated on Drawing No. .... has been submitted by .... (Name of Maker) for test to prove compliance with the definition of flame-proof enclosure (B.E.S.A. Publication No. 229--1926.) and has been found to satisfy the requirements in all respects.

A full report of the tests carried out has been furnished to the maker.

Signed....................(testing authority).

Date......................

Type Tests

6. It is not intended, nor is it recommended, that the tests referred to above shall be made on every piece of apparatus supplied.

Unless otherwise specified when inviting tenders, the purchaser shall accept, as evidence of compliance of the apparatus with this Specification, type tests on apparatus identical in all essential details with the one purchased.

Certificates and full report of all type tests with certified detailed drawings of the type apparatus shall be held available by the maker, together with a record of any alterations, whether essential or not, which have been made to the apparatus since any type test was carried out.

Type tests shall be made by a recognized authority.

Twenty-five Government Departments and Scientific and Industrial Organizations were officially represented upon the Committees entrusted with the preparation of the Specification.

TESTING OF FLAME-PROOF EQUIPMENT

Although the Mines Department tests flame safety lamps, electric lamps, signals, telephones, and shot-firing equipment, it does not test electric motors and their accessories. Such tests as are made are either conducted by the manufacturers of the equipment or are arranged for by them.

The University of Sheffield has an agreement with the manufacturers whereby tests are made of flame-proof equipment at Sheffield. These tests may be witnessed by a representative of the manufacturer of the equipment. Equipment that satisfies the conditions laid down for the tests is given a certificate by the university.

The writer witnessed some of the testing work at Sheffield and was deeply interested in it because similar testing work was being done at the Pittsburgh Experiment Station of the U. S. Bureau of Mines.

The test procedure at Sheffield is as follows:

The submittor of the equipment is required to furnish blue-print copies of drawings showing the general construction of the apparatus submitted. These prints are consulted in deciding whether or not the construction of the apparatus is satisfactory and are further used in checking the dimensions of the various parts making up the equipment. Prints are held on file as a part of the permanent record of the test.

The tests are made in a wooden gallery. The gas used for the explosive mixture is methane, which is obtained from a mine and kept in cylinders. An explosive mixture is kept ready mixed in a large container for certain of the tests. In the case of a motor the first test is made by filling the motor with an explosive mixture of methane and igniting the mixture. During this test one end of the gallery is open. This test is made without an explosive mixture surrounding the motor and is for the purpose of observing whether flames come through any of the flanges or the other joints. After the completion of this test, the front of the gallery is put in place and five additional tests are made with the motor surrounded with an explosive mixture of methane and air. Two of the tests are with the motor at rest and three tests with the motor running. Owing to the construction of the gallery the motor is not under observation in any of the tests in which the motor is surrounded by the explosive mixture. The evidence of safety is therefore based upon the one test made at the beginning under observation and the fact that in the additional five tests the gas was not exploded or the compartment blown apart. Pressure records and analyses of gas samples are obtained for all the explosion tests. The proper proportion of the mixture surrounding the motor is judged by exploding samples drawn from the gallery during the mixing of the methane and air. The methane is slowly admitted from one of the tanks and mixed with a fan as it enters.

The gallery is provided with a partition to conserve the amount of gas necessary for producing an explosive mixture. If the apparatus is small enough it is installed in a half section of the gallery.

A typical certificate follows:

[illegible]

[illegible]

[illegible]

[illegible]

[illegible]

University of Sheffield

MINING DEPARTMENT,

UNIVERSITY OF SHEFFIELD,

(SEAL)

ST GEORGE'S SQUARE,

SHEFFIELD

Telephone 4705

October 15th., 1923.

Certificate No. 40

This is to certify that a typical example of MESSRS. THE ELECTRO-MECHANICAL BRAKE COMPANY, LIMITED, MINING TYPE FLAME PROOF CONTROLLER TYPE 50A., 30. FP., has been treated as follows:-

> The casing of the controller was filled with the most explosive mixture of fire damp and air, and this mixture was ignited by a secondary discharge from an induction coil, whilst the apparatus assembled as for use was surrounded by a similar explosive mixture.

Under these conditions of test, flame did not pass from the apparatus to the explosive atmosphere outside, which remained unignited, nor did the apparatus suffer damage due to the pressure developed within it.

Signed: D. Hay

Professor of Mining.

When one considers the rigid regulations covering electrical equipment in British mines, it would be natural to think that the protective requirements for gassy parts of a mine would be especially severe, but this is not necessarily the case. Much of the equipment used at the face workings has not been tested at Sheffield. The Bureau of Mines has found in testing a great many outfits representing the product of several manufacturers that tests are very valuable in showing unsuspected weaknesses of equipment which might easily be overlooked or not evident from an inspection.

RESEARCH WORK BY THE SAFETY IN MINES RESEARCH BOARD

During a period of several years a systematic study has been made by the Safety in Mines Research Board, under R. V. Wheeler, of fundamentals that may have a direct or indirect bearing on the design of permissible electrical accessories for motor-operated outfits; the following reports have been issued:

1. Flame-Proof Electrical Apparatus for use in Coal Mines, by I. C. E. Statham and R. V. Wheeler, Paper No. 5, First Report, Flange Protection, 1924.

2. Flame-Proof Electrical Apparatus for use in Coal Mines, by C. S. W. Grice, and R. V. Wheeler, Paper No. 21, Second Report, Perforated Protection, 1926.

3. Flame-Proof Electrical Apparatus for Use in Coal Mines, by H. Rainsford and R. V. Wheeler, Paper No. 35, Third Report, Ring-Relief Protection, 1927.

4. The Pressures Produced on Blowing Electric Fuses, by G. Allsop and R. V. Wheeler, Paper No. 38, 1927.

The writer saw most of the equipment used in conducting these researches and conferred with several of the investigators who had been connected with the work. It may be mentioned here that these researches cover work for which the United States Bureau of Mines has never had sufficient personnel, and as the work has been done in a thorough manner it will probably never be necessary for the Bureau to undertake it.

ELECTRICAL INSPECTION

The Coal Mines Act (1911) places the whole responsibility for the selection of suitable equipment, and for its subsequent maintenance in a safe condition, upon the mine owner and his employees; the Mines Department is not charged with the duty of approving electrical apparatus in general. The Inspector of a Division is, however, invested with the power of objecting to the use of electricity in any part of the mine where, on account of the risk of explosion of gas or coal-dust, the use of electricity might be dangerous to life. His objection is subject to appeal to an independent arbitrator.

[illegible]

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[illegible]

[illegible]

[illegible]

The electrical inspection work in Great Britain is in the hands of the Divisional, Assistant, and Junior Mine Inspectors; in case of an electrical accident or a difficult electrical problem, these men call on the Electrical Inspector, who has his headquarters at the Main Office of the Mines Department.

In addition to the Government inspection, many of the owners carry out very complete and systematic periodic inspections. One company with four collieries had a force of 16 electricians and a most elaborate inspection system.

ELECTRICAL EQUIPMENTS IN GREAT BRITAIN AND THE UNITED STATES CONTRASTED

There are a number of differences between the electrical installations in Great Britain and the United States. For instance, in British coal mines there are no trolley locomotives, whereas statistics compiled for 1924 give 11,986 in the United States. Every piece of apparatus and practically every conductor in British mines is earthed by carrying a ground conductor to a ground plate on the surface; practically no earthing is resorted to in American mines except to connect the frames of stationary motors to a pipe or rail return within the mine. Alternating current is not used extensively in American mines, but in British mines this prevails, and direct-current circuits are being replaced by alternating-current circuits in a number of mines. The natural conditions in British mines as to grades, faults, thinness of seams, extreme depth of shafts, and the difficulty of properly supporting the overburden render the installation of electrical equipment much more difficult than in American mines.

In regard to electrical safety it can be said without danger of contradiction that the regulations governing the installation, inspection, and maintenance of electrical equipment are better in Great Britain than in America. In regard to the practice of using strictly safe equipment in gassy sections of mines, there is much yet to be done in both countries.

* * * * * * *

Circular No. 6135
May, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SAFEGUARDING ELECTRICAL EQUIPMENT USED IN GASSY MINES[1]
EUROPEAN PRACTICE: II - BELGIUM

By L. C. Ilsley[2]

INTRODUCTION

Cooperation between the United States Bureau of Mines and the Safety in Mines Research Board of Great Britain, continuous since 1924, has made possible this and other papers on safety subjects. Grateful acknowledgment is made to representatives of the Safety in Mines Research Board of the Mines Department for their assistance in arranging visits to several mine safety stations, and to F. R. Wynne, Deputy Chief Inspector of Mines, Great Britain, for arranging visits to mines in Great Britain, Belgium, Germany, and France.

During the summer of 1927 the writer had the privilege of visiting the mine safety testing stations in Great Britain, Belgium, Germany, and France, in the order named. All of these countries have large coal mines, many of which are rated as gassy. Therefore, when installation of electrical equipment is contemplated, each of these countries is confronted with the same safety problem as is the United States - the development of electrical equipment that will not ignite gassy mine atmospheres which might, through neglect or accident surround the equipment. The means used by these countries in safeguarding gassy mines should therefore be of general interest to safety engineers of American coal mines, and it is proposed to give a brief survey for each of the four countries mentioned. This paper presents the second of these surveys, which covers conditions in Belgium.

CONDITIONS IN BELGIUM

The first paper of the series on electrical equipment in Europe outlined conditions pertaining to coal mines in Great Britain. This second paper gives a similar study of electrical safety work connected with Belgian mining.

As yet Belgian companies have not installed a large amount of electrical equipment in mines, but undoubtedly such equipment will be increased from year to year. However, in the case of gassy mines electrical installations will undoubtedly be under rigid regulations.

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1 The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be used: "This paper represents work done under a cooperative agreement between the U.S. Bureau of Mines and the Safety in Mines Research Board of Great Britain. Printed by permission of the Director, U.S. Bureau of Mines. (Not subject to copyright.)"

2 Electrical engineer, U. S. Bureau of Mines.

Some of the Belgian mines are notably "fiery," and others are subject to instantaneous outbursts of gas; hence that country, for self-protection, has for a great many years given safety engineering a prominent place in its mining work, and consequently the electrical regulations should be of real interest to those confronted with similar dangerous conditions.

CLASSIFICATION OF MINES

In order to have a clearer picture of Belgian mining conditions it is necessary to note their scheme for classifying coal mines. This scheme, which at first may seem complicated, is based on principles of safety, and it would be well if more mining men would think of the possible dangers of their mines from a like angle. The classification follows:

1. <u>From point of view of fire damp</u>:

   (a) Nongassy mines.
   (b) Gassy mines.
      (1) Slightly gassy.
      (2) Gassy.
      (3) Mines subject to instantaneous outbursts.

2. <u>From point of view of dust</u>:

   (a) In nongassy mines, seams of coal containing from 15 to 22 per cent volatile matter (ash free), and in which the brushing of the sides of the roadways puts carbonaceous dust into suspension in the atmosphere, come, as regards regulations applicable to explosives, under, the rules applicable to mines classified in category b (1), that is, slightly gassy.

   (b) In nongassy mines and in mines coming under b (1) owing to fire damp content, the seams of coal containing more than 22 per cent volatile matter (ash free), and in which the clearing of the sides of the roadways puts coal-dust in suspension in the atmosphere, come under the rules applicable to mines classified in category b, (2) that is, gassy.

Nevertheless, if it is established that certain beds give rise to dust which is not dangerous on account of its physical state, classification of the beds under category b (1) can be demanded.

3. The classification of seams of coal can be made for each district or group of districts by the Divisional Inspector of Mines. The classification can be modified at any time. Samples of dust will be collected by the Inspector of Mines and the analysis made by the Mines Department at the expense of the owner.

In mines where naked lights are allowed, it is understood that open-type motors and accessories have been permitted if provided with protective covers. In gassy and slightly gassy mines, approved electrical equipment has been allowed. In mines subject to instantaneous outbursts of gas, no electrical apparatus has been permitted.

TENTATIVE REQUIREMENTS FOR EXPLOSION-PROOF ENCLOSURES

In a pamphlet entitled "Reports on the Problems of the National Institute of Mines at Frameries During the Year 1926," there is included a tentative schedule of requirements for explosion-proof electrical equipment and an outline of the test procedure by M. Emanuel Lemaire, Deputy Director. A free rendering of this schedule follows:

First there is an introduction to the effect that numerous flame-proof types of electrical equipment were tested in gas during 1926. Following these tests, a tentative set of rules to be observed in the manufacture and use of this equipment was drawn up. These rules were not final. They were drawn up to serve as a basis for the exchange of ideas with the manufacturers and users in order to arrive at a set of rules which would assure safety without undue severity on makers or users. The rules follow:

General

Every electrical apparatus whose own circuits (or those circuits of which it forms a part) include windings is considered to be capable of producing dangerous sparks either as a result of normal operation or through accident.

Consequently all apparatus of this kind which is intended to be used in a place where the possible presence of fire damp is to be feared, must be surrounded with an "explosion-proof" enclosure.

Explosion-proof Enclosures

Under this heading are three types - hermetically sealed enclosures, enclosures with protected openings and enclosures with open joints.

A. Hermetically sealed enclosures.

1. Enclosures with an interior volume greater than 2 liters (0.07 cubic foot) must be able to withstand an internal pressure of 8 kilograms per square centimeter (112.76 pounds per square inch.)

2. Enclosures with an interior volume less than 2 liters must be able to withstand an internal pressure of 6 kilograms per square centimeter (85.3 pounds per square inch.)

B. Enclosures with protected openings.

1. Enclosures with protected opening must be able to withstand the pressure developed by the explosion of a fire damp mixture containing 10 per cent methane.

2. The openings in the enclosure must be protected by building up layers of rigid nonoxidizable metal, not less than 50 millimeters (1.97 inches) in width and not less than 0.5 millimeter (0.02 inch) thick, held apart at a maximum distance of 0.5 millimeter (0.02 inch) by means of spacers running the full width of the layers, and spaced close enough so that the distance between layers can not exceed 0.5 millimeter (0.02 inch) if the layers themselves should be bent.

3. The protective devices must be protected against injury and deterioration.

4. The diaphragms of telephone transmitters and receivers shall be protected by a metallic gauze of nonoxidizable material with a mesh of not less than 144 openings to the square centimeter.

C. Enclosures with open joints.

1. These enclosures must be able to withstand the pressure developed by the explosion of a fire damp mixture containing 10 per cent methane.

2. The depth of the opening in the joint can not exceed 0.5 millimeter (0.02 inch), and means must be provided to make any increase of this depth impossible.

3. The width of the open joint can not be less than 50 millimeters (1.97 inches.)

D. Additional methods.

Other methods of stopping flame can be considered allowable after investigation.

General Requirements

1. Assembling can be accomplished:

(a) By screw joints having at least two complete threads.

(b) With flat joints. The surfaces in contact must be machined. Their width must be at least 25 millimeters

(0.98 inch.) The distance between the inner edge of bolt or screw holes and the inside edge of the inclosure must be at least 10 millimeters (0.39 inch).

(c) With flat joints fitted one in the other. The depth of the part fitting into the other must be at least 10 millimeters (0.39 inch). The contact surfaces of the fitted joint must be machined and the clearance between these surfaces must not exceed 0.25 millimeter (0.01 inch).

2. Rubber, asbestos, or other materials of little mechanical strength are forbidden for use in joints except when used in the assembly of fitted-in joints.

3. Bolts, tapped holes, etc., must not pass from one side to the other of the walls of the compartment but must be embedded in sheaths or bottomed holes.

4. Bolts, screws, etc., holding the covers and various parts shall be furnished with devices which will allow them to be loosened only by the use of special wrenches.

5. Shafts, push buttons, etc., must pass through the walls of the enclosure in close-fitting metallic sheaths or bearings not less than 25 millimeters (0.98 inch) in length. The grease grooves must not form a path from the inside of the enclosure to the outside. They must be separated by at least 10 millimeters (0.39 inch), and if they are longitudinal there must be a break at least 10 millimeters (0.39 inch) long.

6. Motor shafts shall pass through the walls of the enclosure in metal bushings at least 50 millimeters long (1.97 inches) for straight bushings. Labyrinthine bushings may be permitted after investigation. The clearance between the shaft and the bushings shall not exceed 0.5 millimeter (0.02 inch). For motors whose air gap exceeds 0.5 millimeter (0.02 inch) the necessary steps must be taken to avoid friction of the shaft on the bushings. Grease cups must be outside the enclosure.

7. Connection of armored cables to the apparatus shall be made:

   1. In a box filled with insulating compound and fastened on the outside of the apparatus. The metal armor of the cable shall be securely attached to the box. The passage of the electric conductors through the wall of the enclosure shall be made through terminal posts firmly attached to the wall and insulated therefrom. On both sides of the wall the conductors shall be connected to the terminal posts by solder or by clamps with screws and lock nuts.

   2. By passing the cable through the wall of the enclosure in a stuffing box with metallic packing in accordance with the following conditions:

(a) The metal armor of the cable shall be firmly attached to the outside of the enclosure in a way that the armor itself shall take any strain to which the cable might be subjected.

(b) The lead sheathing of the cable shall be made bare where it goes through the stuffing box and also, if need be, shall be made cylindrical in shape.

(c) The metal packing of the stuffing box shall be at least 12 millimeters (0.47 inch) long when compressed. The gland nut or gland shall engage in the stuffing box either by screw threads with at least two complete threads engaged or by a fitted joint. The depth of joint or engagement of the parts shall be at least 12 millimeters (0.47) after the packing is compressed.

8. The connection of flexible cables to apparatus must be made in a manner similar to those means prescribed for the armored cables, and to means approved after investigation.

9. The peepholes in the walls of the enclosure shall be protected by a double-thickness glass window solidly set in the walls.

10. Connection to the current supply by means of plugs shall be made as follows:

(a) The socket shall end in a metallic sheath in which is fitted a piece of metal shaped like the plug. At the instant the conducting parts come into contact, the depth of the joint shall be not less than 50 millimeters (1.97 inches). The clearance between the contact surfaces of the joint can not exceed 0.25 millimeter (0.01 inch).

(b) A locking device shall prevent making the socket or plug alive before all parts are assembled and manipulating them when they are alive.

Tests

Each type of apparatus submitted for investigation by the National Institute of Mines shall be submitted to a test consisting of causing an explosion of a mixture of air and fire damp inside the enclosure, with the apparatus in an atmosphere of the same mixture. The enclosure must withstand the pressure of the explosion and the explosion must not be propagated to the outside. This test will be repeated five times.

Ventilated motors shall be kept running for an hour in a current of fire damp and air with candles burning constantly at different points within the enclosure.

There can be no permanent combustion of the fire damp in the interior of the apparatus during the test.

Assembling and Disassembling in the Workings

In underground workings the enclosures of the explosion-proof electrical apparatus can not be opened, disassembled or reassembled except by a person specially authorized by the mine management. This person shall be the sole person permitted to carry the necessary special wrenches. He must supervise the correct assembly, and the fastening of all bolts, screws, nuts, and connection.

The authorized person himself is forbidden to open the enclosure of an apparatus while it is alive, or to make alive an apparatus of which the enclosure is open.

In the program of investigations for 1927, the best way of putting this scheme of regulation into effect is to be determined.

EXPERIMENT STATIONS IN BELGIUM

On August 29, 1927, accompanied by G. Allsop of the British Safety in Mines Research Station and J. A. B. Horsley, H. M. Electrical Inspector of Mines, Great Britain, the writer had the pleasure of visiting the Belgian testing stations at Frameries and Paturage. Frameries has been a noted mine experiment station for a number of years, but in the near future it will be abandoned and all testing work formerly carried on at this station will be conducted close by at Paturage, where there is ample space and where a fine beginning has already been made for a large experiment station.

At the time of the visit the flame-lamp testing gallery, also the gallery in which electric motors are tested, was still at Frameries, and M. Lemaire gave demonstrations with both galleries.

Testing Electrical Equipment

Electrical apparatus is not accepted for test unless it meets the regulations of 1926, already quoted. Drawings are required from the tentative submittor. If these are found to be satisfactory, the equipment can be forwarded for test.

The testing gallery consists of a cubical tank with an opening in the top closed off with paper, and with a glass window for observing the apparatus during tests.

The test procedure is, in general, the same as that set forth in the schedule of 1926. The gas mixtures are made by passing a measured volume of methane into the tank. No analyses are made of the mixture. When making a long test the mixture might become vitiated, so it is changed to offset the effect of the burning of the gas.

The mixture inside the flame-proof compartment is fired by an electric spark. In the case of motors equipped with relief devices, the tests are repeated several times, the motor being allowed to run several hours. If the mixture external to the motor is not ignited by these internal explosions, the motor is judged to have demonstrated its safety. No records are taken of the explosion pressures developed.

CONDITIONS IN BELGIUM AND THE UNITED STATES CONTRASTED

In Belgian mines nearly all of the electrical equipment is of the alternating-current type, whereas in the United States only a small proportion of the underground mining equipment is of that type. The large use of alternating equipment in Belgian mines is probably due to the great depth of the mines (in some cases from three to four thousand feet); if low-voltage, direct-current systems were used a large outlay for copper conductors would be required. By using alternating current the electrical energy can be taken down the shafts at a high potential and transformed inside the mine to a working potential.

In most Belgian mines, because of their great depth, it would be difficult to maintain a roof that would offer satisfactory protection to a trolley locomotive system. Although the haulageways might be arched over with brick or concrete, squeezes would undoubtedly occur that would require extensive and frequent repairs. Hence, even if the gas conditions permit, it is not likely that trolley locomotives will ever be used to any large extent in Belgian mines. However, the use of battery locomotives may be considerably extended. Cheaper labor, small cars, and constricted roadways have a tendency to keep back the introduction of mechanical equipment.

If all electrical equipment used in gassy mines is examined and listed in accordance with the specifications of 1926, a high order of safety ought to result as compared with conditions in America, where each of the 30 coal-mining States makes it own regulations and where various degrees of safety prevail depending upon the State law and the strictness of State enforcement officers.

* * * * * * *

Circular 6136 May, 1929

INFORMATION CIRCULAR

Department of Commerce -- Bureau of Mines

PROGRESS IN METAL MINE VENTILATION

BY

D. Harrington

Circular No. 6136
May, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

PROGRESS IN METAL MINE VENTILATION[1]

By D. Harrington[2]

Correct ventilation of underground workings whether in coal or in metal mines consists in establishing such control of air currents that the miners may work in safety, with maximum comfort and efficiency, and without impairment of health; also in controlling the air flow so that the mine workings may at ordinary times be kept reasonably free of harmful gases or dusts, and so that in time of emergency, as when a fire or an explosion occurs, there may with minimum delay be maintained as much or as little air flow as is desired throughout the mine in its entirety or in certain parts of the mine.

Control of air flow is the essential point of any mine ventilating system, and is obtained only by the installation of mechanically operated fans, and other ventilating devices such as doors, overcasts, regulators, etc. Every mine, large or small, coal or metal, should from the outset be equipped with a fan. Much has been written about natural ventilation, and many claims have been made that in specific mines or parts of mines there is sufficient natural air flow. As a matter of fact, there are actually very few if any mines, coal or metal, where natural ventilation creates atmospheres anything like adequately safe or healthful for underground workers even at ordinary times or under ordinary conditions; and at the time of a fire or an explosion mines depending upon natural ventilation are practically helpless, and are decidedly dangerous for the unfortunates forced to be in them either at the time of the disaster or afterwards when trying to handle conditions arising from the emergency.

Coal mines are usually compelled by State law or by existance of explosive gas to establish some sort of ventilating system, but most metal mines rarely pay any particular attention to air flow until forced to do so by some unusual circumstance or by unfavorable underground working conditions. State or other laws as to ventilation of metal mines are practically nonexistent, and the few laws on the statute books of the various States of the United States concerning ventilation of metal mines are essentially meaningless; at best they, like the State laws as to coal mines, are woefully crude and inadequate. Notwithstanding this, there is annually some progress made in metal-mine ventilation, and the year 1928 has produced at least a slight amount of data which should be of aid in forwarding ventilation betterments in metal mines.

1 This paper was presented at the February, 1929, meeting of the American Institute of Mining and Metallurgical Engineers. The Bureau of Mines will welcome reprinting, but requests that the following footnote acknowledgment be used: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

2 Chief Engineer, Safety Division, U. S. Bureau of Mines.

Cooling of Mine Air

Much has been written during 1928 concerning the many problems associated with the desirability, in some cases the absolute necessity, of cooling the air in working places in some of the most productive mines - gold, silver, copper, iron, and other metal mines, as well as coal mines. As in the past, most of the data concerning developments in the cooling of mine air have come directly or indirectly from foreign countries and particularly from South Africa.

The following abstract from the 1927 Report of the Government Mining Engineer of South Africa indicates that in mines which are rated as hot, not only the efficiency but also to an increasing extent the very lives of the underground workers must depend upon the finding of some efficient method of cooling air.

> Deaths from heat stroke for the year 1927 totalled 8, as compared with 6 during the latter portion of 1926. This class of accident was first included in the accident statistics in August, 1926, when it was decided that such casualties were directly due to the conditions under which the persons affected had to carry on their occupation, and were likely to increase as the mines became deeper.
>
> As the workings have become deeper and the rock temperature higher, it has become apparent that more elaborate and systematic ventilation arrangements are necessary. It has always been known that the efficiency of the worker becomes less as the temperature and humidity increase, and now that deaths have occurred amongst some workmen and others have been temporarily incapacitated, the urgency for improved ventilation and cooling of working places has been realized. It is evident that, unless this matter is dealt with satisfactorily in the future, the decreased efficiency of the workers will result in an increase of working costs and consequent restriction of operations.
>
> It is also generally recognized that, as the temperature and humidity increase, the hazard of accident becomes greater, owing to lowered vitality of the individual and dulled reaction to a sense of danger.
>
> Serious attention is now being given to the above matter, and notable improvements in some of the mines are already apparent.

In the same report, pages 66 and 67, are the following details concerning some of the essential conditions surrounding the workers, and also a description of the working conditions where the cases of fatal heat stroke occurred.

> Four deaths of natives from heat apoplexy were reported, all from the City Deep. In the previous year there were seven such deaths, of which six were at the Village Deep and one at the City Deep.

The fatal cases at the City Deep were all among natives who were not thoroughly acclimatized. Two of the deceased were working their second shifts in the mine; one was working his third shift in the mine and his first at the place where he was overcome; and one was working his fifth shift in the mine and his first at the place where he was overcome.

If the record is examined in another way, it is found that two were working in probationary gangs, and one had just been removed at his own request from a probationary gang in which he had worked four days, as a few days before he had been working underground at another mine. One of the deceased had not been placed in a probationary gang. His case was the first during the year, when the organization for acclimatization was faulty.

Records supplied by the mine, showing approximately what the conditions were at three of the places where the fatalities were caused, may be placed on record. They are--

(a)	Kata-thermometer ...	11.0	wet	3.88	dry
	Hygrometer	86.5° F.	wet	87.0° F.	dry
(b)	Kata-thermometer ...	9.65	wet	2.29	dry
	Hygrometer	84.6° F.	wet	85.2° F.	dry
(c)	Kata-thermometer ...	5.4	wet		
	Hygrometer	90.0° F.	wet	90.5° F.	dry

Two of the deaths occurred in March, one in April, and one in November.

The absence of deaths at the Village Deep during 1927 was doubtless in part due to the precautions outlined in my last annual report, which include probationary or acclimatization gangs for new natives, salt in drinking water, and meals underground; and perhaps also was partially due to injections of a preparation named "Lobelin" when symptoms of danger were noticed, but it was mainly due to the improved ventilation produced by the more powerful ventilating fan installed during 1926.

Six nonfatal cases of collapse due to heat were reported from the City Deep, and two such cases from the Village Deep. The cases at the City Deep included an instructor and four apprentices in the Government Miners' Training School section. These are the only cases among Europeans yet reported. It was ascertained that each of the five persons was in either the inception or convalescent stage of mumps at the time of his collapse.

From the foregoing description it would appear that the fatalities occurred in practically saturated air with temperatures between 85 and 91° F.; such temperature and humidity conditions are to be found in a considerable percentage of the working faces in many of the deep metal mines of the United States.

In the Mining Magazine of London of September 28, 1928, is the following description of the use of ice in cooling working places in the Boulder Perseverance gold mine of Kalgoorlie, Australia, the ice supposedly serving the double purpose of cooling the working place and of aiding in some manner in the allaying of the fumes and possibly the dust left in the air from blasting:

Ice in Blasting.- In the Kalgoorlie Mine for July 4, Ernest Williams, manager of the Boulder Perseverance gold mine, gives some particulars of his experience with ice for cooling mine air at the time of blasting for the purpose of accelerating the removal of fumes from the atmosphere.

Fumes and high temperatures have always been sources of trouble and expense in mining development work at Kalgoorlie. Mr. Williams and his staff have evolved a new scheme, the use of ice, to overcome these drawbacks, and the successful results achieved by them will materially reduce the cost of winzing and work in development ends. In May, 1927, Mr. Williams, in order to reduce the amount of time lost in waiting for fumes to clear from winzes (sometimes up to two hours in deep winzes) tried the device of throwing ice into the winze to clear the atmosphere. Finding that the ice had the desired effect, he developed the idea and the present practice is to break up about 50 lb. of ice into 6 in. pieces and throw it down immediately after firing. The cold condenses the fumes and forces out the hot air and the men are able to return to the bottom of the winze 40 minutes after firing. As far as it is known, ice has never been used in this manner before, though ice has been used in South Africa and parts of America to cool the main air currents in the ventilation system.

The procedure in using ice in drives or cross-cuts is somewhat different. This scheme was first tried in September, 1927, and after a time it was found that the most successful method was to place the ice in a box, specially designed, and connect up a 2 in. compressed-air pipe. The ice-box holds approximately 60 lb. of ice when charged and 1 cwt. of ice is consumed in a shift. The ice-box is placed about 80 ft. away from the face and the cool air is taken through an outlet pipe and discharged 15 to 20 ft. from the face. As soon as the fuses are lighted the cool air is turned on, and 30 minutes after firing the atmosphere is clear, and the wet and dry bulb temperatures well below the mark set by the Arbitration Court. The margin between the wet and dry figures, as shown in the accompanying table, indicates the effect of this use of ice. The table is a record of temperatures taken in advancing four cuts at the 1,100 ft. level of the Perseverance on the X lode north drive. The readings were taken between May 24 and June 4, 1928. The ice-box was used only during firing. The temperatures before firing were taken with the icebox shut off. Each cut was approximately 5 ft. and 30 minutes after firing the atmosphere was clear. Temperatures as shown in the table were then taken and work was resumed.

TEMPERATURE FAHRENHEIT

	Before firing		First firing		Second firing		Third firing		Fourth firing		Fifth firing	
No. of cut	Dry	Wet	Dry	Wet	Dry	Wet	Dry	Wet	Dry	Wet	Dry	Wet
No. 1 Face	77½	75¼	76½	73½	78½	75	78	73½	78	73½	--	--
No. 2 Face	76½	73½	76	73	77	74	78	74	78	74	--	--
No. 3 Face	76	73	77½	72½	78	73	78	73½	78	72	--	--
No. 3 Face	78	71½	--	--	--	--	--	--	--	--	76	70
No. 4 Face	78	75	78	74	77	73½	77	70	77	71	78	71

Apparently metal mining companies in South Africa and Australia are using ice to a considerable extent with the idea of making underground temperatures endurable for workers in contact with hot rock.

In the South African Mining & Engineering Journal, issue of August 18, 1928, was presented the following rather optimistic forecast as to a probably effective method of locally cooling hot humid air in deep working faces:

As anticipated in these columns some months ago, a solution of the problem of working in high temperatures in the deep level mines of the Rand seems now in sight owing to the successful research work done by some of our leading engineers. The invention in question is a portable air cooler which claims to be an economic solution of the difficulties met with in a hot humid atmosphere. The mines on the Rand are particularly free from natural, dangerous or poisonous gases, and, with modern ventilating methods and single-shift working, bacteriological or chemical contamination is practically unknown. We have only to contend against heat and humidity, because, outside these factors, the air is good and suitable. The problem, therefore, resolves itself into cooling the air at any particular working place and extracting the moisture. The underlying idea in the new apparatus is that certain sections of a mine are treated as independent units. The ordinary ventilating current in each section of convenient size is improved by removing moisture and cooling the air by taking advantage of the cheapest available sources of power and cooling mediums and utilizing these in a combination of machines and devices, which can serve any selected section. When a section is worked out, the machine can be removed to another section. The principles involved are:--1. The compression of warm, saturated mine air sucked into the machine from any section of the mine. 2. The cooling of the compressed air to about its initial temperature by circulating mine water. 3. The extraction of the moisture condensed during the cooling process whilst at high pressure. 4. The expansion of the high pressure air in an air motor so that a low final temperature is obtained and the dry air discharged to working places. The power developed by the air motor is utilized to assist in driving the compressor. 5. Reheating the cold expanded air by stages, so as to prevent troubles due to very low temperatures and to reduce the power consumption by utilizing the circulating water passing from the coolers during or after compression of air. 6. The balance of power required to drive the compressor not furnished by the air motor and making up frictional and other losses to be supplied by an electric motor. Such a machine as herein outlined has not yet been built, but it seems fairly evident that it offers a plausible and possible solution of the heat and humidity problem in ultra deep-level mining, because of three prominent factors. 1. Cheap electrical power supply available underground. 2. The production of localized cool and dry air without any undue heating of the mine, because a comparatively small quantity of water is required and warmed up but a few degrees, so that in nearly all cases the seepage water in the mine (which must in any case be pumped to the surface) will suffice to cool and dehumidify the air at selected places with very little extra expense in pumping. 3. It is also probable that the capital cost will be moderate, as it

is not necessary to dehumidify and recondition the full ventilating current of a mine, but only to deal with the air used in the sections where active work is being carried on. In a recent discussion on the subject, Mr. W. Elsdon Dew said that in an endeavor to analyze the theoretical considerations and to put to practical use the proposed methods, an enquiry had been issued to certain manufacturers who would be able to design and manufacture a machine to suit the conditions laid down. If a suitable machine at a suitable price could be provided, a practical test could then be carried out. The experiment was worth a trial, and only by such an experiment could progress be made. The industry as a whole is, of course, vitally interested in the matter, and we hope to chronicle at a later date the progress being made.

In the United States attention is being more definitely directed to the problem of mine-air cooling and conditioning. The following paragraph from the National Coal Association Bulletin of September 15, 1928, brings to the fore some underlying principles of refrigeration which in some form or other may be applicable to cooling mine air:

> Attention is directed to a new type of iceless refrigerator cars, the operating principle of which is based partly on the compressed gas system used in lighting passenger coaches. A supply of gas is carried in fuel tanks under the car. Burners are lighted and are extinguished automatically at intervals. A hard, glassy substance, with the appearance of clear quartz and capable of absorbing a large amount of moisture is used. This material is called silica gel and is heated by the burners. The ceiling of the car is piped with sulphur dioxide as the refrigerant. It is circulated by the physical action of the absorption and the driving out of the vapors from the silica gel. The first twenty-five of these cars are in service. It is stated the temperature can be set and maintained for an entire trip, with but little variation. Thus we have the forerunner of what may prove to be a revolution in refrigerator-car service.

There also appeared the following item in the September 4, 1928, issue of Power, published by the McGraw-Hill Publishing Co.; here again is described a method of air conditioning for offices in surface buildings which might with suitable modification be applied to the local conditioning of air at faces in mines having hot and humid but otherwise pure atmosphere:

> It is stated that a Carrier air washer and a Brunswick and Kroschel 3-ton carbon dioxide refrigerating machine and other equipment were installed to ventilate an office which might contain about 60 men and women, and in order to secure the delivery of 30 cu. ft. of air per minute per person a fan with a capacity of 2,000 cu. ft. of air per minute was installed. The air was passed vertically through a fine spray of water, which, besides cleaning it, cooled it to about 55°F. The consumption of electricity by the refrigerating plant was about 14.3 kw. per hour.

There is no question that cooling of mine air is a live subject in connection with the safe and efficient working of many of our metal mines; although application of some form or forms of local cooling or refrigerating or conditioning of air will undoubtedly be necessary in some mines or under some circumstances, nevertheless the most effective remedy to date for the "conditioning" of air at working faces or places in mines is the continual circulation of relatively large volumes of fresh pure air from the surface to and past the places where men work. This means that the best remedy for the relief of hot humid air in mines is the installation and operation of an adequate ventilation system.

Dust and Dust Disease in Metal Mines

For some years it has been recognized that one of the most effective methods of combatting the dust menace in metal mines is the continual coursing of currents of fresh air to and then away from the places where men work - usually the places where the dust is most harmful in quantity and quality. Although little is heard concerning the health hazard from dust in mines, the hazard continues to exist; and although some alleviating methods and practices have been installed and used in many of our more progressive metal mines - practices such as extension of ventilation, use of wet drilling, use of water sprays, water blasts, etc., shooting largely if not wholly at the end of the shift, etc. - nevertheless both in the United States and in other countries which use much more drastic methods than we do for prevention of mine-air dustiness and of dust disease of mine workers, there continue to be much silicosis and other kindred diseases due largely if not wholly to dust.

There is a well-defined tendency to believe that the only dust which is dangerous to inhale is that which is composed largely of silica, especially of free silica. That this is a dangerous fallacy was plainly shown by a study of the effect of dust inhalation on cement workers as reported in Bulletin 176 of the U.S. Public Health Service in 1928. The results of this study indicated that silica constituted less than 25 per cent of the dust and that of this amount of silica only a very small portion was free silica - the material generally held essential for harmfulness from dust. However, breathing large quantities of the very finely divided dust caused the cement workers to have nearly three times the absentee record from colds, bronchitis, influenze, and grippe; resulted in skin diseases causing five times as much disability; and also brought on diseases of the eyes, ears, pharynx, tonsils, and rheumatism three times as often as among workers of the same age-range in a rubber plant of similar size and number of employees. It was found that a large percentage of the workers voluntarily left the industry because of its ill effect of one nature or other on the health, thus leaving in the industry those whose powers of resistance were inherently higher than the average. It was also found that about one-third of the workers who had been employed three years or longer had one or more respiratory defects or chronic respiratory diseases. Of 37 X-rays taken, 15 showed dust disease of the lungs, 3 showed tuberculosis, and 8 others showed more or less evidence of tuberculosis. It was definitely concluded that one of the main problems of the cement industry is dust control; this decision is the more significant from the viewpoint of the

metal-mining man in that the dust had little or no free silica and that much more than 50 per cent of the dust was lime, which has heretofore been considered by many authorities to be harmless and by others to have remedial rather than harmful effect when breathed from the air. The application of this conclusion to the mining industry means that any or all finely divided dusts in the air and in large quantities when breathed for considerable periods daily are likely, in fact almost certain, to be harmful; hence all dusty methods, processes, or practices in metal mining should be eliminated or at least limited. The best methods of doing this are the exclusive use of wet drills in drilling, the postponement of blasting of all kinds until the end of the shift or after the regular shift, and the use of continuous air currents to and past working regions or other dust-producing places to keep the finely divided dust out of the areas where men must work.

In view of the long-continued systematic and expensive fight waged against silicosis and other dust diseases in the metal mines of South Africa, the following editorial which appeared in the South African Mining and Engineering Journal on August 25, 1928, is of more than usual interest; the article indicates that even when virtually all known methods of prevention are placed in effect, many of them of a far more drastic nature than even thought of in the United States, the dust diseases (silicosis, tuberculosis, etc.) are by no means eliminated.

> We have not got rid of silicosis, but I believe, as I have said, that we have turned a big corner in the matter. Dr. Irvine draws attention to the different way in which Europeans and natives are affected by the two risks, dust and tuberculous infection. "Of all cases of compensatable disease--under which term one includes the three clinically distinguishable conditions of "simple silicosis," "tuberculosis with silicosis," and "simple tuberculosis,"--of all such cases which occur in European mines, over 88 per cent are at the outset cases of clinically simple silicosis. A little over 6 per cent only have active simple tuberculosis or obvious tuberculosis along with silicosis when first detected. These are the data for the last two years taken as a whole. Amongst mine natives, on the other hand, the position is reversed. Over 83 per cent of all cases of compensatable disease in natives are obviously tuberculous, and more than half of the total are cases of active simple tuberculosis. Simple silicosis accounts for only 16 per cent. There are also data for the last two years. Compared with what we know of European communities, the prevalence of tuberculosis amongst the mine natives is not exceedingly high. But, in the gross, cases of active simple tuberculosis or of active tuberculosis with silicosis totalled in the last year nearly 1,200, each a potential source of infection of others and of European miners. What one feels is that miners' phthisis has, in the past, been apt to be viewed by technical men too much as simply a dust question, and not enough as a question of tuberculosis." The necessity for the fullest possible control of native tuberculosis is stressed but the author concludes: "I feel more hopeful about miners' phthisis than I have felt for many a long day. But that hope should encourage us all the more to push on with the job, so that all the time we may do better and better. With increasing depths and temperatures the problem is not growing easier. But people look to us here on the Rand for instruction in the best practical

methods for the control of this industrial disease which is a menace to many mining communities the world over. And they are coming here in 1930 to learn how."

Mine Fires

Metal-mine ventilation and metal-mine fires are so closely related that any extended reference to one of them almost necessarily brings the other into the discussion. The Hollinger fire on February 10, 1928, in the great Porcupine gold mining district of northern Ontario was the outstanding metal-mine fire of the year, and was also the first extensive metal-mine fire in point of loss of life in the history of metal mining in Canada; 39 lives were lost, though the property loss was trivial. The fire stunned the metal mining men of Ontario, as they had been of the opinion that these mines were essentially immune to fire. This attitude is only too prevalent among metal mining men throughout the world; not until it is realized that any and all metal mines, even those which are essentially wet, have a definite fire hazard will there be a cessation of the metal-mine fires causing loss of life out of all proportion to the amount of the fire itself.

Practically all metal mines have combustible matter in and about them in the form of timber, explosives, oil, etc., which when burning may readily give off fumes of such poisonous nature as to snuff out lives by the score. The material which burned and gave off poisonous fumes sufficient to kill 39 underground workers was chiefly débris from explosives boxes, sawdust, etc., that had been thrown into a nearby stope - a practice which has been more or less prevalent (and one whose probable dangers would be, in fact have been, on numerous occasions, scornfully minimized) in metal mines not only in Canada but in various parts of the United States. The total amount of combustible matter was small, but the fumes given off in the confined passages of the mine were deadly. In 1928 a fire in a metal mine in Mexico resulted in the death of over 20 men, yet the total amount of charred underground mine timbers (little or none of them absolutely burned) amounted to less than 3 cords.

Unfortunately practically all metal mines have in them not only enough combustible matter to burn with attendant heavy loss of life under some circumstances, but also practically all have readily available the flame to ignite the combustible matter. The igniting flame is generally from open lights, almost in universal use in metal mines, from tobacco, matches, and other smoking materials, and from electricity; the dangers from arcing and sparking are intensified through the unsafe electrical installations and practices almost universally prevalent in metal as well as in coal mines. Also the numerous (almost innumerable) dangerous practices in handling explosives, including dynamite, fuse, and detonators, in metal mines give an almost ever-present source of flame for ignition of any incombustible matter which may be within reach.

Probably the Hollinger fire with its 39 fatalities was started by an open light; the Granite Mountain fire in Butte, Mont., in 1917 with 163 fatalities was caused by an open light; the Magma fire in 1927 with 7 fatalities was probably also caused by an open light, possibly a cigaret; and there have been numerous

other metal-mine fires in late years due to open lights or smoking. In fact, open lights have been the igniting cause of more metal-mine fires during the past 10 years than any agency except electricity; and there will continue to be periodical metal-mine fires with disastrous loss of life until metal mining men realize that open lights not only are dangerous but also are less efficient, less safe, and less economical than the safe-sane up-to-date types of storage battery electric cap lamps.

Subsequent to the Hollinger disaster Judge Godson, who was appointed by the Province of Ontario to conduct an inquiry into the disaster, made two reports, and in the final one he made a number of recommendations after having taken volumes of testimony from miners and mining men not only at the Hollinger mine but also at other representative properties of the Province. The recommendations of Judge Godson in a report dated September 28, 1928, are as follows:

I recommend that the Mining Act of Ontario governing the operation of mines be amended by varying or adding thereto in substance the following submissions:-

1. That every man employed as an underground foreman, (meaning thereby one who is exclusively engaged in supervising the work of other men), shall be able to give and receive orders in the English language.

2. That an Inspector of Mines shall have the right to suspend any foreman or mine captain who is not familiar with or does not understand the requirements of the regulations governing the operation of mines as contained in the Mining Act of Ontario.

3. That the words 'above ground' in the first line of Section 161, Sub-section 11, of the Mining Act be deleted and the Section read as follows:-

   "No building for thawing explosives shall be maintained in connection with any mine except with the written permission of the Inspector of Mines. The site of this building and the style of structure and equipment shall be subject to the approval of the Inspector. The building shall be under the direction of the manager or some person authorized by him. The quantity of explosives brought into any thawing house at any one time shall not exceed the requirements of the mine for a period of twenty-four hours, plus the amount that it may be necessary to have thawing to maintain that supply."

4. That all underground structures necessary for the installation, maintenance and repair of machinery and equipment should be fire proofed.

5. That all fans except "Booster" fans should be placed on the surface and be reversible, and all underground fans should be in fire proofed housing.

6. That oil and grease kept underground be contained in suitable metal receptacles and should not exceed one week's supply.

7. That there should be a sufficient number of fire doors at every station where practicable, so that the shaft could be completely cut off from the rest of the mine.

8. That all inflammable waste or rubbish should be taken to the surface.

9. That shift bosses and mine captains should certify at least once a week that there is no accumulation of combustible waste or rubbish underground, except as noted, in the area under their supervision.

10. That rescue stations be located at a place selected by the Chief Inspector of Mines in the Timmins, Kirkland Lake and Sudbury mining areas, and be in charge of one man to be appointed by and under the control and direction of the Department of Mines. It should be the duty of such employee to take care of the apparatus, train men in the mines in his area in rescue work and inspect and report upon the apparatus, if any, maintained at any such mine.

11. That each rescue station should contain the following or other equipment to be ultimately determined:-

- 1 Tool Chest.
- 15 Oxygen cylinders - 100 cubic feet each.
- 1 Portable Orsat apparatus for making analysis of mine air.
- 1 Anemometer for measuring ventilation.
- 1 Psychrometer for determining humidity of mine air.
- 1 Maximum and Minimum thermometer.
- 2 Cabinets (first aid) with extra bandages and splints.
- 4 Canaries for testing mine air for carbon monoxide.
- 2 Stretchers.
- 12 Self contained oxygen breathing apparatus with accessories for testing, repairing and recharging.
- 1 Pyrotannic acid detector for determining carbon monoxide in blood and air.

5 All-service gas masks with extra canisters.
1 Iodine pentoxide detector for indicating amount of carbon monoxide in the air of the mine.
1 Geophone.
1 Oxygen inhaler for administering oxygen in conjunction with artificial respiration.
1 Oxygen pump for re-charging small tanks for breathing apparatus.
1 Lifeline 1200 feet used for rescue crews when exploring mines after fires or explosions.
12 Electric cap lamps with accessories and charging equipment.
12 Approved-type flashlights.
20 Bottles for collecting samples of mine air.
Cardoxide.

The above equipment was suggested and put before the Commission by the Chief Inspector of Mines at the inquiry held at Haileybury. He was not then able, however, to definitely say it should be adopted in its entirety. It should be at once reviewed by the Inspector and the Committee representing the operators and finally determined. The equipment adopted should be used in all stations so that there would be uniformity.

12. That fire protection systems be installed at all underground crushers, tipples and in dry shafts.

13. That for the purpose of a uniform danger alarm all mines in Ontario should have equipment for pumping into air lines a stench chemical to be selected and adopted by the Chief Inspector of Mines.

14. That readable signs showing the way to emergency exits should be posted in prominent places underground and all men should be instructed where these emergency exits are placed.

15. That the Chief Inspector of Mines may order an underground connection be made between adjoining mines where he deems it necessary for the safety and protection of underground employees.

"This proposed regulation invades the right of ownership, may involve an expense largely for the benefit of an adjoining property and otherwise open up contentious questions. While I deem it expedient to recommend it as a safeguard in a remote but possible contingency, there should be a proviso allowing the right of appeal from the order of the Inspector to a person or tribunal to be decided upon.

Gases Encountered In Metal Mines

During the year it has been amply demonstrated that metal mines continue to encounter numerous dangerous gases; in fact, metal mines appear to have by far a greater variety of dangerous gases than are to be found in coal mines, notwithstanding well-defined opinion to the contrary.

In the Mining Magazine of London, issue of August, 1928, there is an abstract of an article entitled "Natural Gas in Mines on The Rand," and the following tabulation taken from that abstract gives some analyses by the Government Laboratory of South Africa of gases found in the City Deep, Robinson Deep, and Crown mines, all famous for gold production.

	Crown Mines, %	Robinson Deep, %	City Deep, %
Hydrogen	77.5	15.4	24.0
Methane	11	55.2	5.0
CO_2	0.1	0.2	- -
Oxygen	0.3	6.0	- -
Inert (Nitrogen etc)	11.1 (difference)	- -	- -
CO	- -	Nil	Nil

The gases tabulated are of particular interest because of the exceptionally large percentage of hydrogen in all of them, but particularly in the one from the Crown mines. The combination of methane and hydrogen gives a decidedly inflammable mixture, and it is fortunate that large quantities of these gases are not found.

These gas occurrences are described in more detail in the following abstract taken from pages 67 and 68 of the 1927 Annual Report of the Government Mining Engineer of The Union of South Africa:

> Three occurrences of inflammable gas in workings at great depths were reported during the year from the City Deep, Robinson Deep and Crown mines respectively. They indicate that an unexpected form of danger is to attend future deep mining on the Central Rand, but whether or not it is to be one of great importance remains to be seen. According to present indications the ventilation of the lower levels necessary to dilute and carry away the dust thrown into the air by mining operations and to cool the workmen will be sufficient, except in rare cases, to deal with the gas encountered.
>
> In the City Deep on 9th November, 1926, inflammable gas was found issuing from a shot-hole socket in a winze below the thirty-second level, No. 4 (a) shaft, at a depth of about 6,746 feet below the surface. The gas caused bad headaches, and, when lighted, burnt strongly with a blue flame.

It is reported to have shown, on qualitative analysis, the presence of carbon monoxide and methane.

In another winze on the same level, but 325 feet away, a greater volume of gas was encountered on 18th February, 1927. The gas, which bubbled up on the footwall side of the winze, burnt with a strong blue flame. Analysis of a sample gave 24 per cent hydrogen, 5 per cent methane, and carbon monoxide, nil.

Both these occurrences in the City Deep were in shale below the reef horizon. The first was 460 feet and the second 280 feet distant on the normal from the Leader.

In the Robinson Deep on 20th September, 1927, gas was found issuing from a shot-hole socket in the face of 4,400 East Drive, Alma shaft. When ignited it burnt with a flame about 18 inches long, the flame being yellow tinged with blue. Analysis showed the gas to contain methane, 55.2 per cent; hydrogen 15.4 per cent; oxygen 6 per cent; carbonic acid 0.2 per cent; and carbon monoxide, nil. The place of occurrence is in quartzite 5,341 feet below the surface, at an unknown distance, probably about 50 feet normal, below the Leader, and about 280 feet normal above the bed of shale in which the City Deep occurrences were found.

In the Crown Mines on 18th November, 1927, a small explosion burnt two natives who were working a machine drill in the face of No. 29 Haulage, East Drive, No. 15a shaft. The gas continued to issue from the hole they had been drilling, and to burn with a roaring flame 5 to 6 feet long. It continued to burn till 24th November, the flame having then diminished to a few inches in length. A sample of the gas taken on that day gave an analysis, methane 11 per cent, hydrogen 77.5 per cent, carbon dioxide 0.1 per cent, oxygen 0.3 per cent, and inert gases (nitrogen, etc.) 11.1 per cent. Carbon monoxide and olefines were absent. When work was continued, with precautions, inflammable gas continued to be found issuing from various drill holes and slips in the rock for several days. The place of occurrence of the accident was in quartzite about 4,675 feet below the surface, about 230 feet normal below the Main Reef Leader, and about 140 feet normal above the shale bed.

It seems to be generally assumed, as is not improbable, that the inflammable gas has its origin in the shale bed, though only the occurrences at the City Deep were actually in that shale.

The large percentages of hydrogen found on analysis are remarkable.

No occurrences of inflammable gas have yet been reported from the Village Deep, which is noteworthy, as that mine lies between the Robinson Deep and the City Deep, and has extensive development workings below the Main Reef Leader at depths considerably exceeding that of the occurrence in the Robinson Deep, and slightly exceeding those of the occurrences in the City Deep.

A novel gas occurrence in some Colorado oil wells is described in the National Petroleum News issue of July 18, 1928, in which a gas, mainly carbon dioxide, was encountered in enormous quantities and under enormous pressures. One well yielded approximately 40,000,000 cubic feet of the gas per day, and the pressure was so great that it was impossible to keep drilling tools in the well in order to drill deep enough to reach the desired oil strata, supposed to be some hundreds of feet deeper. The gas on escaping absorbed so much heat that practically everything would soon be frozen solid near the top of the well or in the region for hundreds of feet below the top. These frozen bridges at times "let go" or are made to "let go" by drilling with the usual tools; but when the release occurs it is stated that "the tools, or bailer, or whatever happens to be in the hole leave for other parts, and so does the crew." So far as known, this gas has not yet been put to any useful purpose.

During the year a fire occurred due to methane ignition in a wet shaft, part of the Hetch Hetchy water project, near San Francisco; notwithstanding the extreme wetness of the shaft, methane "feeders" which had become ignited continued to burn and finally ignited some of the timber; the flames were later extinguished, however, without loss of life. Partly as a result of this affair the following orders were issued by the California Industrial Accident Commission in October, 1928:

1. Adequate ventilation shall be provided at all times in all underground workings in order to prevent explosive gases from collecting.

2. No open lights, smoking, matches or smoking material shall be allowed in underground workings. A large notice to this effect shall be posted in plain view at each headframe and portal.

3. Lighting conductors underground shall conform to Orders 713-7 (h) and (i) of the Electrical Safety Orders of the Division of Industrial Accidents and Safety.

4. Immediately before shooting, tests of the air with a flame safety lamp of a type approved by the United States Bureau of Mines shall be made by a competent, designated person, and if the presence of explosive gas is indicated by this lamp, no shots shall be fired until the workings are free from gas. Men shall not be allowed to go to work after blasting until tests have been made, and the workings found to be free from all gas.

   Whenever the tunnel operations are not continuously prosecuted, immediately before resuming work, a test of the air shall be made with the flame safety lamp, and the men shall not go to work until the underground workings are found free from gas.

5. If, for any reason, ventilation should be interrupted, continuous tests of the air shall be made, and if the presence of explosive gas is indicated by the flame safety lamp, then all men shall be removed to the surface until such time as ventilation has been restored, and no presence of gas is shown by the safety lamp.

All men shall be removed to the surface in instances where the ventilation has been interrupted for more than sixty minutes and not returned to work until the flame safety lamp shows that no gas is present.

6. A ladder of sufficient length shall be provided in each heading for the man making tests, so that he may be able to test the air throughout the workings.

7. These orders shall not apply to those men specifically designated to make gas tests, restore ventilation or perform other emergency operations under proper supervision.

The foregoing orders constitute an official recognition of the necessity in tunnel work of taking definite precautions against the gas ignitions which have so frequently occurred in tunnels being driven for water, sewer, or other purposes in or adjacent to our large cities. The orders should be extended to include prohibition of bare power wires, open types of electrical switches or electrical pumps, locomotives, etc., in any tunnel where carbonaceous shale, slate, etc., or coal seams may be encountered; strata of this kind, especially when below the water level, are very likely to give off explosive gas and in some instances poisonous gases such as hydrogen sulphide.

Sealed fires, usually of spontaneous origin, from time to time give trouble in some of the iron mines of Michigan where carbonaceous shales or slates have been encountered. These fires are difficult to handle, and the general practice is to attempt to seal the affected region; occasionally, however, the seals break or become crushed and thus release the gases from the sealed regions so that the air of surrounding workings becomes contaminated. The escaping gases frequently contain appreciable percentages of the actively poisonous hydrogen sulphide and of the somewhat less dangerous sulphur dioxide, with carbon dioxide in the amount of 5 per cent or over and oxygen reduced to 12 per cent or less. In some instances high temperatures are found, but the temperature is usually less than 200° F.

CONCLUSION

Although our metal mining people are slow to realize the benefits of ventilation, there are occasional indications that at least some of our wide-awake operators realize the benefit of expending a reasonable amount of time, attention, and money in the effort to secure adequate air circulation in their mines and definite control of that circulation. C. F. Kelley, president of the Anaconda Copper Mining Co., in 1928 made the following statement:

I can remember when members of our staff said the final limit of this operation will not be the continuity of mineral content but the impossibility of ventilating and securing air under which men can live; I can remember when the budget was put up $3,000,000 to ventilate the Butte mines in a single lump. Had it not been for the organization that could carry that through upon a unified system, this operation would cease to exist.

Inasmuch as the Anaconda Company continues to ventilate and even to expand in its ventilation work, it is fair to assume that this company has learned pretty definitely that ventilation pays.

The following paragraph occurs in a paper by A. C. Butterworth, electrical engineer, mining department, Pickands, Mather and Co., presented before the American Society of Mechanical Engineers at St. Paul, Minn., in August, 1928:

> Partly because of a growing consciousness of the average unsatisfactory air conditions underground, more and more attention is being paid to the question of mechanical ventilation of the mines. There is not the necessity of getting rid of explosive gases which exists in some coal mines, but the heat and the carbon dioxide due to decaying timber matting, together with the powder smoke underground, make it necessary to pay attention to this problem. In the early days many mines had reasonably good natural ventilation, but with the increased depth of mining even those mines are now turning to mechanical ventilation. Many devices have been tried out to improve local ventilation in exceptionally poor places. Direct blowing of compressed air and the use of air "injectors" and air-operated fans and blowers have generally given way to the use of the latest type of portable motor-driven ventilating fans. The trend of this work, however, is toward the large mine-ventilating unit, together with a proper system of air doors throughout the mine, to give adequate and positive ventilation everywhere in the mine. At some mines it is found necessary to operate these big fans only occasionally, but sometimes conditions make it necessary to run them continuously; and without question, considerably more power can be used to good advantage for this purpose in most underground mines in this territory. Although no comparative figures are available, it stands to reason that fresh air will keep the mining efficiency higher than if the air were hot and dead, as is frequently the case at present. At one property operated by Pickands, Mather & Co., where the fan is used continuously, the power consumption is about 1 kw.-hr. per ton; and, as expressed by the underground men, 'this fan is worth a million dollars.'

One of the mine operating officials of Pickands, Mather & Co. expresses his opinion as follows:

The principal advantage of mechanical ventilation to the operator has been the increased efficiency of the miners and in the saving of time to the men thru being able to return to his working place in a shorter time after each blast.

With the improved condition underground, it is easier to maintain a full force at all times. For example, the average temperature was reduced about four degrees and the relative humidity lowered between 3 and 4 per cent. The more rapid removal of smoke and gas has helped considerably, but no doubt the greatest benefit has been derived thru lower temperatures and reduced humidity. In some places where the temperature was lowered only one or two degrees, the rapid moving air made working conditions more comfortable and have resulted in a more efficient operation. This improvement in working conditions has materially increased the efficiency of the miners thru a saving in time to the miners by the quick removal of smoke and gas and a marked reduction in temperature and relative humidity.

J. B. Pullen, ventilation engineer of the Phelps Dodge Corporation, in a paper in The Mining Congress Journal for September, 1928, describes ventilation practice in the Phelps Dodge mines at Bisbee, Ariz., and concludes as follows:

> It is impossible to show in dollars and cents just what benefits the company has derived from the installation of mechanical ventilation. Nevertheless it is a recognized fact that certain sections of the mine could not be worked without it. Also, by bringing about such improved conditions it is natural to assume that the efficiency of the underground men must have increased to some extent. Aside from this, good working conditions will attract good men, making it easier to build up a productive organization. This alone is certain to have a favorable effect on costs.

In concluding this report, it seems desirable to call attention to the fact that many interesting points have been brought out in a paper by O. A. Glaeser[3] concerning the ventilation of the United Verde mine at Jerome, Ariz. There is no question that the ventilation system in the United Verde mine is one of the best to be found in metal mining.

3 Proceedings of the February, 1929 meeting of the American Institute of Mining and Metallurgical Engineers.

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CIRCULAR 6137

MAY, 1929

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

WORK OF THE HOLMES SAFETY ASSOCIATION IN THE STATE OF WASHINGTON

BY

J. G. SCHONING

Circular No. 6137
May, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

WORK OF THE HOLMES SAFETY ASSOCIATION IN THE STATE OF WASHINGTON[1]

By J. G. Schoning[2]

INTRODUCTION

The Holmes Safety Association is an offspring of the Joseph A. Holmes Safety Association which was organized in Washington, D. C., in 1916 at a meeting of representatives of twenty-four leading mining associations connected with every branch of the United States mineral industries. The purpose of the meeting was to do honor to the memory of Joseph A. Holmes, one of the founders of the United States Bureau of Mines and its first director, whose death had occurred the year before on July 12, 1915.

In deference to what was believed would have been the expressed wish of this great humanitarian and benefactor of the mineral industries, it was decided to establish a national safety organization to cover the mineral industries exclusively; out of respect for his memory and service, the organization was named the Joseph A. Holmes Safety Association. On receiving requests from several mining districts, in 1922 a number of chapters were established in several mining States.

In 1926 authority was granted to establish the Holmes Safety Association to absorb the existing chapters and promote the present chapter plan with self-governing authority and representation on the board of directors of the Joseph A. Holmes Safety Association.

OBJECTS AND SCOPE OF THE HOLMES SAFETY ASSOCIATION

The Holmes Safety Association has filled a definite want; heretofore, no nation-wide safety organization existed with the main object of promoting safety in mining, and every branch of the mineral industries, with abnormally high fatality and injury rates involving the safety of more than 1,000,000 employees, has greatly needed such an organized safety movement.

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1 - The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be used: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

2 - Foreman miner, U. S. Bureau of Mines Mine Safety Station, University of Washington, Seattle Wash.

Another advantage of the Holmes Safety Association is that it is national in scope, covers every branch of the mineral industries, and has the endorsement and support of numerous existing organizations and associations of employees connected therewith. It has cooperative relations with similar safety organizations in other industries, has affiliations with the National Safety Council, and is an integral part of the great American safety movement.

OFFICIAL MONTHLY PUBLICATION

The Holmes Safety Chapter Notes published monthly by the association provides a medium through which chapter officers and members can keep informed on the progress of chapter activities elsewhere and is also a means of disseminating useful and helpful information on safety and other subjects of interest. This publication, which is small and in its infancy, is the only paper of its kind devoted exclusively to the promotion of health and safety in the mineral industries of the United States. As the Holmes Safety Association advances and spreads its influence, the Chapter Notes will undoubtedly grow accordingly in size and usefulness.

SAFETY WORK IN THE STATE OF WASHINGTON

The broad scope of the safety, educational training, and welfare activities of the association was found to be especially applicable to the isolated coal-mining communities of the State of Washington, and chapters were established in mining communities of leading coal companies in 1922 and 1923.

Mining Regulations of Washington

The mining regulations for the State of Washington require a mine safety organization and stipulate that safety committees in which representatives of the employees participate shall make a bimonthly inspection of the mine with the object of remedying any existing hazards that may be discovered. It is further required that the safety committees shall report to general monthly safety meetings in which all employees are required to participate.

Failure on the part of the mining company to hold such meetings and to submit a report on the same is evidence that the safety educational standards of the State industrial insurance law have not been complied with, and under such circumstances no refund is made to the employing company for reductions effected in fatality and injury rates.

The Holmes Safety Association is admirably suited to meet the requirements referred to. It supplements the work of the mine safety organization, provides a means for complete organized safety effort on the part of all mine officials and their employees and families, and supervises the regular monthly safety meetings, which are held under the auspices of the local chapters.

Coal Production

Washington is classed among the small coal-producing States; in 1927 the mines produced between two and three million tons and employed a total of 3,330 inside and outside workers.

Fatality Rates

The fatality rate for Washington compared with the average rates for bituminous mines in the United States for the past five years is given in the following table.

Number Killed per Thousand 300-day Workers

	1923	1924	1925	1926	1927
Washington	6.53	10.01	13.74	6.31	9.21
United States, bituminous.	4.65	5.39	4.79	4.86	4.60

It is generally conceded by mining experts that Washington has some of the most hazardous bituminous mines in the country, because of the steeply pitching seams, bad roof, and other unusual hazards. The State of Washington fatality rates are abnormally high; however, the rates at the mines where Holmes safety chapters are conducted efficiently are far below the State average.

Organization

Listed below are the seven local chapters of the Holmes Safety Association which have been organized in the State of Washington for the purpose of promoting the safety movement among employees in the mines.

Name of company	Name of chapter	No. of chapter	Location of chapter	Date of organization
Abandoned	Elk	23	Palmer	Oct. 5, 1922
Bellingham Coal Mines	Bellingham	53	Bellingham	Apr. 7, 1923
Abandoned	Burnett	55	Burnett	May 1, 1923
Pacific Coast Coal Co.	Carbonado	56	Carbonado	May 1, 1923
Do.	New Castle	70	New Castle	Sept.21,1923
Do.	D.C.Botting	72	Black Diamond	Sept.29,1923
Washington Union Coal Co.	Tono	170	Tono	- - -

Of the seven chapters organized in the State, two of them, No. 23 at Palmer and No. 55 at Burnett, have passed out of existence, as the mines at these places have been worked out and closed.

During the time the Elk Safety Chapter was functioning, it was actively engaged in safety work under the able direction of Mr. M. C. Butler.

The mines at Carbonado, New Black Diamond, and New Castle are operated by the Pacific Coast Coal Co. In 1922 the company organized what is known as the Pacific Coast Coal Company Mine Council, with local councils at the mines which meet monthly, and a central council composed of representatives of all mines which meets in Seattle once each month. A meeting of all supervising officials is held once each month at each mine, at which time accidents that have occurred since the last meeting are brought up for discussion. In addition to these, the monthly safety meeting is held as required by the State mining law. The State safety meeting and the Holmes Safety Chapter meeting are held in conjunction with each other. All members of the mine council and inspection committee must be members of the Holmes Safety Association.

Bellingham Safety Chapter No. 53 is active, although there is no suitable place to hold meetings, so that the work of the chapter has been somewhat handicapped. The Bellingham Coal Mines, Inc., at whose mine this chapter was organized, contemplates erecting a building for the purpose of holding safety meetings, and also for the holding of first-aid and mine-rescue practice. Mr. James Pascoe, superintendent of the Bellingham mines, says, "The Holmes safety chapter has been helpful in reducing accidents, and the safety instructions received at the chapter meetings have been effective. Though there are still some men who are prone to take a chance, the great majority keep before them the words 'Safety First.' There is a safety committee man on each level in the mine and two on the outside, the first mechanic and electrician. These committee men, meet once each month, make bimonthly examinations of all inside working places, and interview the workmen as to measures being taken for their safety. The committee also visits all men and working places on the outside. During its existence this chapter has held many interesting and instructive meetings. When available, capable speakers are obtained from without as well as from within the industry, and at the conclusion of the meeting entertainment and sometimes a substantial lunch is provided by the company.

Tono Safety Chapter No. 170 has accomplished much good by its safety work. The Washington Union Coal Co. has erected a neat community club house in which the chapter holds its meetings once each month. The program consists of regular business, safety talks and discussions, and general educational and safety features which serve to initiate and maintain interest. Following the usual business meeting, a talk on safety is given by an outside speaker. The safety director reads clippings from papers covering different types of accidents and comments on them, then a discussion is held by the members. Following this, the educational director reads a short biography of some noted person. A short period is devoted to first-aid demonstration by two or more members of the gathering, after which a lunch is served. The ladies furnish the lunch one month, and the men the next; and the men not only serve it but do all of the cooking and baking of the cakes and pies. The excellence of the lunches served by the men as well as those furnished by the ladies is known to all who have participated. Mr. Pontin and several of the men's first-aid team have donated much of their time in giving first aid training to the boys in the school; and Mrs. Way and some of the ladies have done the same for the school girls.

The chapter meetings are a clearing house for discussions of unsafe and safe practices. The members report unsafe practices observed in the chapter mines, and after thorough consideration, make recommendations for changes necessary to remove the hazards. That the meetings are actively devoted to safety discussions is shown by the following minutes of two of the meetings:

"Carbonado Chapter No. 56, Carbonado, Wash., held its regular monthly meeting October 25, with a very good attendance. Mr. M. A. Morgan, Chairman, opened the meeting and read the minutes of the previous meeting, which were approved. The special features of the meeting were several instructive and interesting talks by W. R. Reese, State mine inspector, who stressed the importance of safety meetings and urged a larger attendance. Mr. Geo. C. Hewitt followed with a talk on 'Safety methods' and urged a better safety record. W. A. Wilson, general superintendent, gave a direct and personal talk on "Safety" and urged the practice of safe methods. Mr. John Wallace, labor commissioner for the Northwestern Improvement Co., gave a very interesting and instructive talk on 'Safety' and vividly portrayed the need of safety methods in industry of all kinds. Several selections were rendered during the evening by the Carbonado community band. The following items of interest regarding safety and health work in the community were taken up and discussed: Taking tools in and out of the mine - special car is now being provided for this purpose; guard rails - guard rails on trolley at chutes will be remedied; ladders in steep pitch - ladders in Morgan and No. 4 seams are too light and stronger ones will be substituted; signals for hoisting timbers - signals in pillar for hoisting of timber are being looked into; placing batteries at face - Mr. McKim spoke of the need of a safe place for men to stand when going back into place after blasting, will be looked into; moving of bull wheel - the new position of bull wheel should be well timbered; opening of crosscuts on west side - this is being done. After a lengthy discussion of the above items, the meeting adjourned.

"Newcastle Chapter No. 70, Newcastle, Washington, held its regular monthly meeting August 15 with a good attendance. The meeting was called to order by chairman Jack Parker and the minutes of the previous meeting were read and approved. The following unfinished business was taken up and discussed. Coal coming down the manways, which was reported as satisfactory. Mr. Scobie, jr., reported that another block will be completed in a week for the extra string of red lights. Another man car has been added to the trip. The men have also been requested to make as little noise on the man trips as possible. First-aid supplies for bunkers are always on hand at the office. The following new business was discussed: Mr. Parker spoke regarding the men not wanting to come to the meetings, stating that their suggestions were both helpful and appreciated in cutting down accidents. Mr. Hewitt stated that a report of each accident should be read and discussed at the meetings and be placed in the minutes. It was suggested by Mr. Parker that no antiseptic dressing should be used by those treating wounds, as it made it easier for the doctor. Mention was made of the bunker boys riding the cars. They were advised to be careful as this practice was very dangerous. Fire bosses were instructed to see that the afternoon shift man trips were side-tracked. Mention was made that foot

posts were too short; these will be ordered longer. A bad hole was reported in the floor of the bunkers. It will be repaired immediately. Mr. Hewitt spoke of the wise choice of our chairman, as Mr. Parker has had considerable to do with the safety work in this State. He also spoke of Mr. Wilson's desire for the betterment of cooperation at these meetings, which will in turn make a better mine and better working conditions. Mr. Parker noted that each man was responsible for his place. He also entreated the men to persuade the other fellows to be careful at all times as accidents happen very easily."

Holding open forums on safety encourages the men to be careful and also to observe closely other men in order to obtain information on unsafe practices that may be reported at the meetings. The results of constant safety thought are shown in decreased accidents.

Community Safety Activities

The chapters have been instrumental in the development of the Boy Scout and Camp Fire Girls organizations, also the development of first aid in the schools. Through the efforts of the chapters 100 per cent enrollment in first-aid training in the schools has been obtained. Chapters take much interest in promoting such activities in the camps. At stated times, general open safety meetings are held which are attended by almost every person in the camp. These meetings, in addition to the ordinary subject presented, provide entertainment of various types. Burnett Chapter No. 55 at Burnett, while in existence, operated on the same plan as the present active chapters.

ACCIDENT RATES

Following are the accident frequency and severity rates of the mines of the Pacific Coast Coal Company from 1924 to 1928, inclusive:

Accident Rates of Pacific Coast Coal Co., 1924-28

Year	Accident-frequency rate[1]	Accident-severity rate[2]
1924	161.01	77.32
1925	109.07	102.88
1926	140.83	94.66
1927	109.37	47.74
1928	153.01	19.77

1/ Frequency = $\frac{\text{(Total number of accidents) (1,000,000)}}{\text{(Total hours worked)}}$

2/ Severity = $\frac{\text{(Total shifts worked) (1,000)}}{\text{(Total hours worked)}}$

Considerable variance will be noted in the frequency rate. This is due to the fact that up until 1928, noncompensable accidents that did not have sufficient loss of time to allow State compensation were not reported. Beginning with 1928, the safety educational work was revamped and required that any

accident causing more than twenty-four hours' time loss should be reported. This would naturally make the 1928 accident-frequency rate considerably higher than for any other year since 1924. However, the severity rate in 1928 is greatly decreased over any previous year listed.

Owing to the fact that no statistical data are available or have been worked out locally by any of the companies having chapters except the Pacific Coast Coal Co., no comparison of frequency and severity rates other than those shown can be given. However, data furnished the State mine inspector and covering the district as a whole show a reduction in both frequency and severity. This is attributed to more closely organized cooperative safety effort on the part of the mine officials and employees.

CHAPTER SUPPORT

The Holmes Safety Association chapters must rely on the active support of the employing companies and company officials for its success. Open approval and support of the chapters by company officials insure the continued active interest of the most capable men in directing its activities so that the maximum results will be obtained in the promotion of health and safety at work, in the homes, and in public places. With proper direction the safety movement will automatically attract enthusiastic support from all sources, for safety is the very basis of our existence.

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Circular 6138 May, 1929

INFORMATION CIRCULAR

Department of Commerce -- Bureau of Mines

METHOD AND COST OF MINING HARD SPECULAR HEMATITE ON THE MARQUETTE RANGE, MICHIGAN

BY

Lucien Eaton

Circular 6138
May, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

METHOD AND COST OF MINING HARD SPECULAR HEMATITE ON THE MARQUETTE RANGE, MICHIGAN [1]

By Lucien Eaton[2]

INTRODUCTION

The mining methods described in this paper are those used in a large mine producing hard specular ore on the Marquette Range in the Upper Peninsula of Michigan. Although the conditions of mining are unique in this district, they are similar to those in many other districts, and information in regard to the methods used and the results obtained should be of interest. The methods now used are an outgrowth of those of former years which have been adapted to conform with improvements in mechanical appliances, changes in the wage scale and labor supply, and the selling price of the product.

The mine is situated at Ishpeming, in the Upper Peninsula of Michigan, 16 miles west of Marquette, its shipping port on Lake Superior.

ACKNOWLEDGMENTS

For much of the information in regard to the early history of the mine the author is indebted to Mr. James E. Jopling, of Marquette, Mich., who has been a mining engineer in the district for over 45 years. Information has also been taken from reports of the U.S. Geological Survey and the Geological Survey of Michigan, from publications of the Lake Superior Mining Institute, and from articles written by the author for the Explosives Engineer and the Engineering and Mining Journal.

HISTORY OF THE DISTRICT

Ore was first discovered on the Marquette Range in September, 1844, by United States surveyors who were making township surveys of the region. The first commercial body of ore was found by Marji Gesick, a chief of the Chippewa tribe of Indians, who showed it to explorers in September, 1845. This discovery was made on what was later the property of the Jackson Iron Co. in the city of Negaunee. In 1850 a large outcrop of ore was discovered in Ishpeming about 2 miles

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2 One of the consulting engineers, U. S. Bureau of Mines.

west of the original discovery. The ore in the mine to be described is an extension of this outcrop, but the fact was unknown for many years. The ore for which the mine was opened was discovered by diamond-drilling nearly 30 years after the outcrop was found.

The first shaft was begun in 1880, and another shaft, 800 feet farther west, was begun in the following year. Both shafts were dropped through quicksand. The second shaft was lost; after it had lain idle for about two years, however, it was recovered by the use of cast-iron tubbing and was straightened and sunk to the ore. With the exception of the years 1893 to 1897 and 13 months in 1921 and 1922 the mine has operated continuously. The total production has been approximately 10,000,000 tons.

Both shafts are vertical, and the ore was hoisted for many years on cages, but in 1910 the cages were superseded by skips of much larger capacity.

GEOLOGY

The ore occurs in the Upper Marquette Series of the Huronian Period, and is overlain by the Ishpeming quartzite and slate, with a footwall of jasper and altered diorite. It occurs as an irregular bed, badly folded and faulted, varying in thickness from a few feet to 100 feet, with an average thickness of 25 to 30 feet in the larger ore bodies (figs. 1 and 2). There is an unconformity between the ore and the hanging wall and probably in many places between the ore and the footwall, although the latter is not easy to determine.

In general the ore occurs in a syncline with an axis running east and west. This axis is not straight but pitches downward to the east and to the west from a high point near the middle of the ore body. Both sides of the syncline are present only near the middle as one side or the other has been removed by erosion or displaced by faulting in the other parts of the mine. Three major longitudinal faults have been recognized, and several minor longitudinal and cross faults are known to exist. The dip of the ore is irregular and in general is too low for the ore to run on the footwall by gravity. This fact and the occurrence of many interbedded seams of rock in the ore have had an important influence on the choice of the method of mining.

The ore formation is overlain by quicksand from 50 to 100 feet deep, and the ore bodies have been followed for long distances under the business portion of the town where the surface rights are not owned by the mining company. For these reasons it is necessary to use a method of mining that does not involve disturbance of the surface.

PHYSICAL CHARACTERISTICS OF ORE AND ENCLOSING ROCKS

The ore is a hard, specular hematite low in silica and moisture and containing appreciable amounts of lime and magnesia. The iron content is high, about 59 per cent natural, the phosphorus content is 0.100 per cent, and the silica is 5 to 7 per cent. It is separated into two grades, "lump" and "crushed." The physical structure, low moisture, and low silica content make the lump ore especially desirable for the open-hearth furnace.

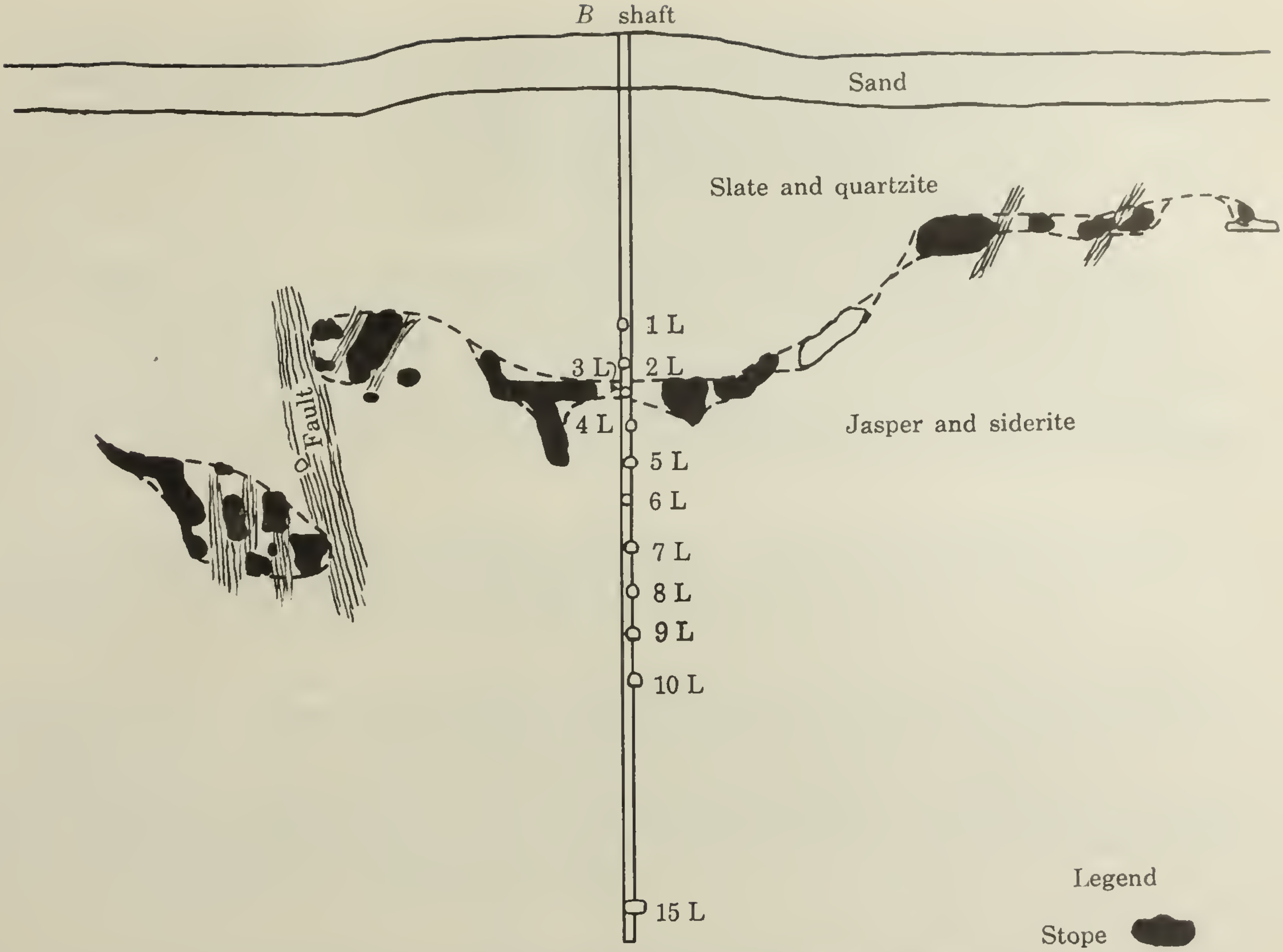

Figure 1. - North - south cross section through shaft, looking east

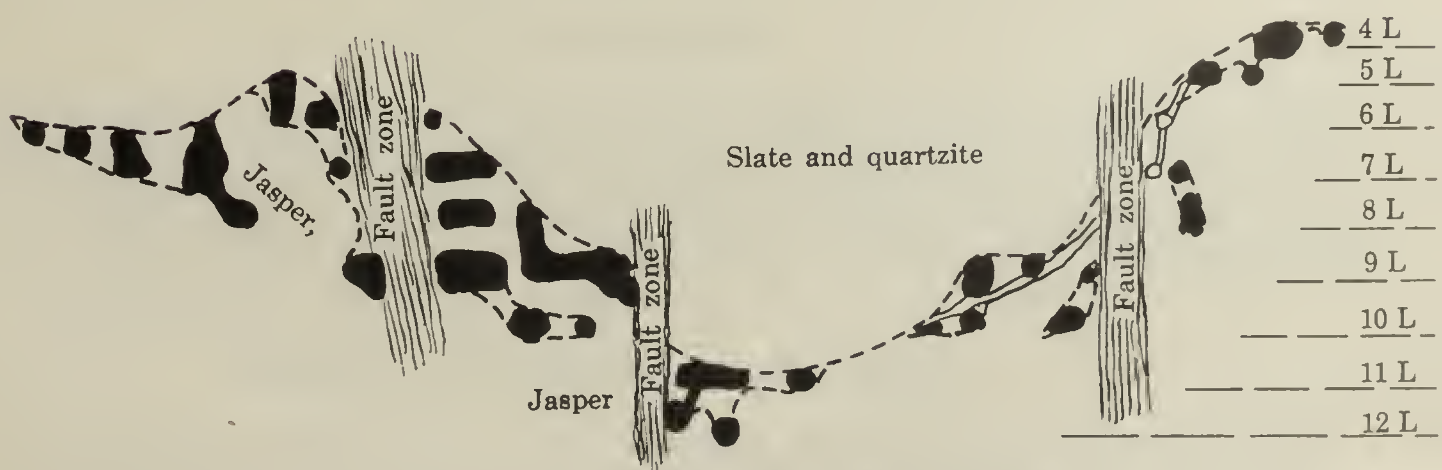

Figure 2. - North - south cross section 2,100 east

The ore is tough and hard to drill, and it stands well on exposure to the air. Seams of rock from 1 or 2 inches up to 1 or 2 feet in thickness often occur in the ore and must be sorted out by hand underground. This fact and the flat dip of many of the ore bodies have made the use of shrinkage stopes impracticable, and a system of open rooms and pillars, with the ultimate recovery of many of the floors, has been found to meet the situation admirably.

The hanging wall in most places is a slate which slacks when exposed to the air; for this reason a small amount of ore is left to protect it wherever possible. Overlying the slate, which is seldom over 15 feet thick and usually much less, there is a strong, hard quartzite which does not weather or swell when exposed to the air, and which forms an excellent capping for the stopes in the system of mining employed. Under the slate in some places there is a conglomerate of jasper and ore.

The footwall is jasper or altered diorite. The jasper stands well under all conditions, but the altered diorite occasionally gives trouble, especially where it lies at a steep angle or has been sheared by faulting.

The displacement of the ore bodies by faulting and their irregular size and occurrence have rendered prospecting difficult and have made it impossible to block out large bodies of ore for mining. Great flexibility in the method of mining is required, and the ability to follow the ore wherever it goes is essential. These two conditions, as well as the inadvisability of caving the surface and the necessity of sorting out waste rock underground, have been determining points in the choice of a mining method.

In many parts of the mine, where there has been a good deal of faulting, a small amount of magnetite has been formed in the hematite mass; this has seriously interfered with the orientation of local strikes and dips and has complicated the study of the geology, especially of the many systems of joint planes.

SHIPPING FACILITIES

The mine is situated 16 miles from Lake Superior and is served by three railroads. Most of the ore is shipped by rail to Marquette and thence by water to Lake Erie ports, but a substantial tonnage is shipped entirely by rail to Canada and to the Chicago and St. Louis districts. The demand of customers for lump ore of high quality and of limited size has had an important bearing on the choice of mining method and on the treatment of the ore on surface.

METHODS OF PROSPECTING AND EXPLORATION

The earliest exploration was by vertical diamond-drill holes from the surface; a great deal of ore has been discovered in this way. In later years the underground workings standing open have become very extensive, so that it is now more economical and more effective to explore by horizontal diamond-drill holes underground. One diamond drill is kept in use nearly all the time, and averages over 2,000 feet of hole per year. Over 400 holes have been drilled underground during the life of the mine.

Drilling is done with a screw-feed drill driven by compressed air, using "E" rods. Two men are required underground and part of a diamond-setter's time is needed on the surface. As the mine is piped with air and water for the rock drills, there is little difficulty in making a set-up anywhere.

A large part of the exploration is by drifts and raises, and the ore bodies are usually developed by breast stopes and raises. The old rule of following the ore is a very valuable one in this mine.

METHODS OF SAMPLING AND ESTIMATING TONNAGE AND VALUES

Production samples are checked by samples taken at intervals from the stopes and drifts and by the appearance of the ore. Only one grade of ore is hoisted, so that the sampling is a simple matter.

The standard methods adopted by the mining companies of the Lake Superior region and by the Lake Erie chemists are used to sample the product on the surface. These methods are fully described in the "Methods of the Chemists of the U.S. Steel Corporation for Sampling and Analysis of Iron and Manganese Ores," third edition, from which pertinent extracts are published in the 1927 edition of Crowell and Murray's "Iron Ores of Lake Superior."

Estimating the tonnage of ore available in the mine is an intricate and arduous task. With a few exceptions the ore bodies are most irregular, and their outline is not fully known until a large part has been extracted. The ore bodies on each level are carefully mapped, and the amount of available ore is calculated as closely as possible. The gross tonnage is also calculated, and the difference between the gross tonnage, the sum of the available ore, and the ore already extracted is the amount left to support the surface. The different pillars and floors are numbered, and a record is kept on special maps which are brought up to date once a year by subtracting the ore mined from and adding the new ore developed to the figures for the previous years. This system is practically a continuous inventory. It is checked by recalculations every few years.

In calculating developed tonnage allowances of 10 per cent for incidental rock and 10 per cent for loss in mining (in this case mostly ore left to protect the hanging-wall slate and in the rounded corners of the arches of the rooms) are deducted from the gross amount.

The analysis of the ore in the different parts of the mine is estimated from composite stope samples for each year and from the results of diamond-drilling allowance is made for contamination by incidental rock.

Values are calculated according to the formula of the Lake Superior Iron Ore Association. These calculations are too intricate to be included here, but may be found in the 1927 edition of Crowell and Murray's "Iron Ores of Lake Superior," page 107 et seq. At the present time a base ore ($51\frac{1}{2}$ per cent iron natural) of non-Bessemer grade is worth $3.25 a ton at the mine. For specific ores the price fluctuates according to analysis.

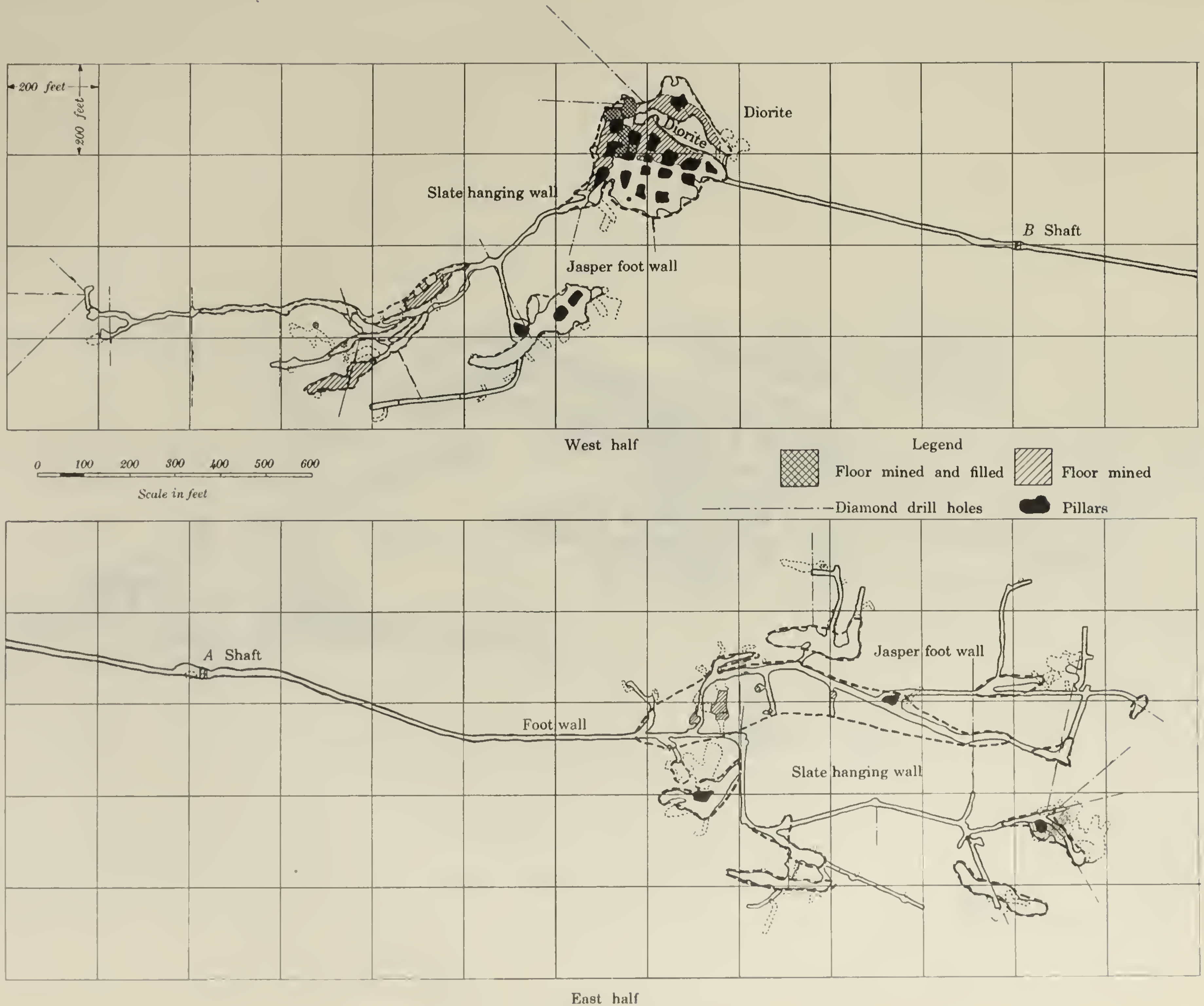

Figure 3.– Outline of ore bodies on tenth level

Figure 4. · Outline of ore bodies on second level

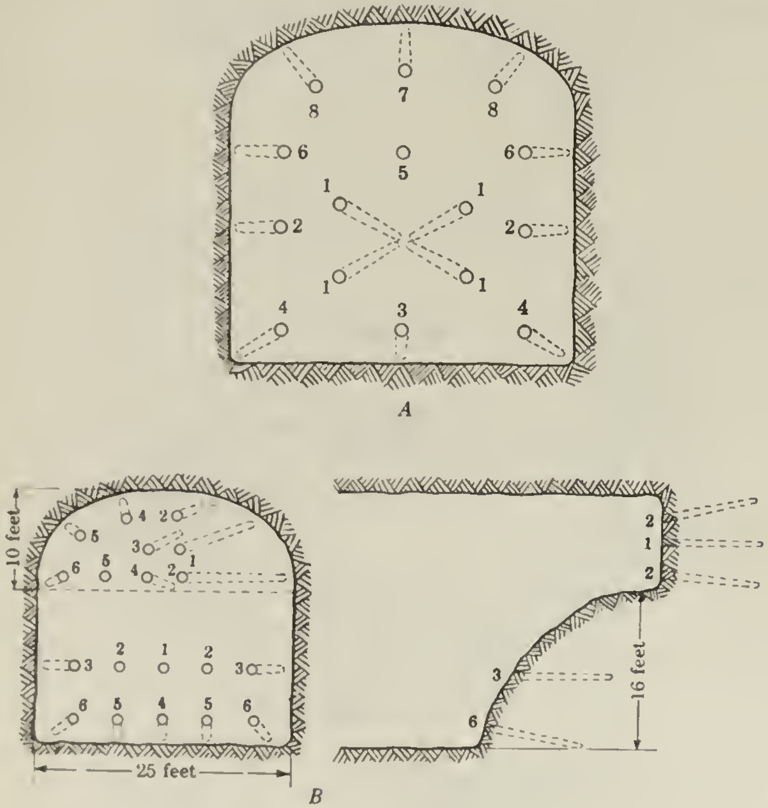

Figure 5.— A, A drift round.
B, A breast stope round

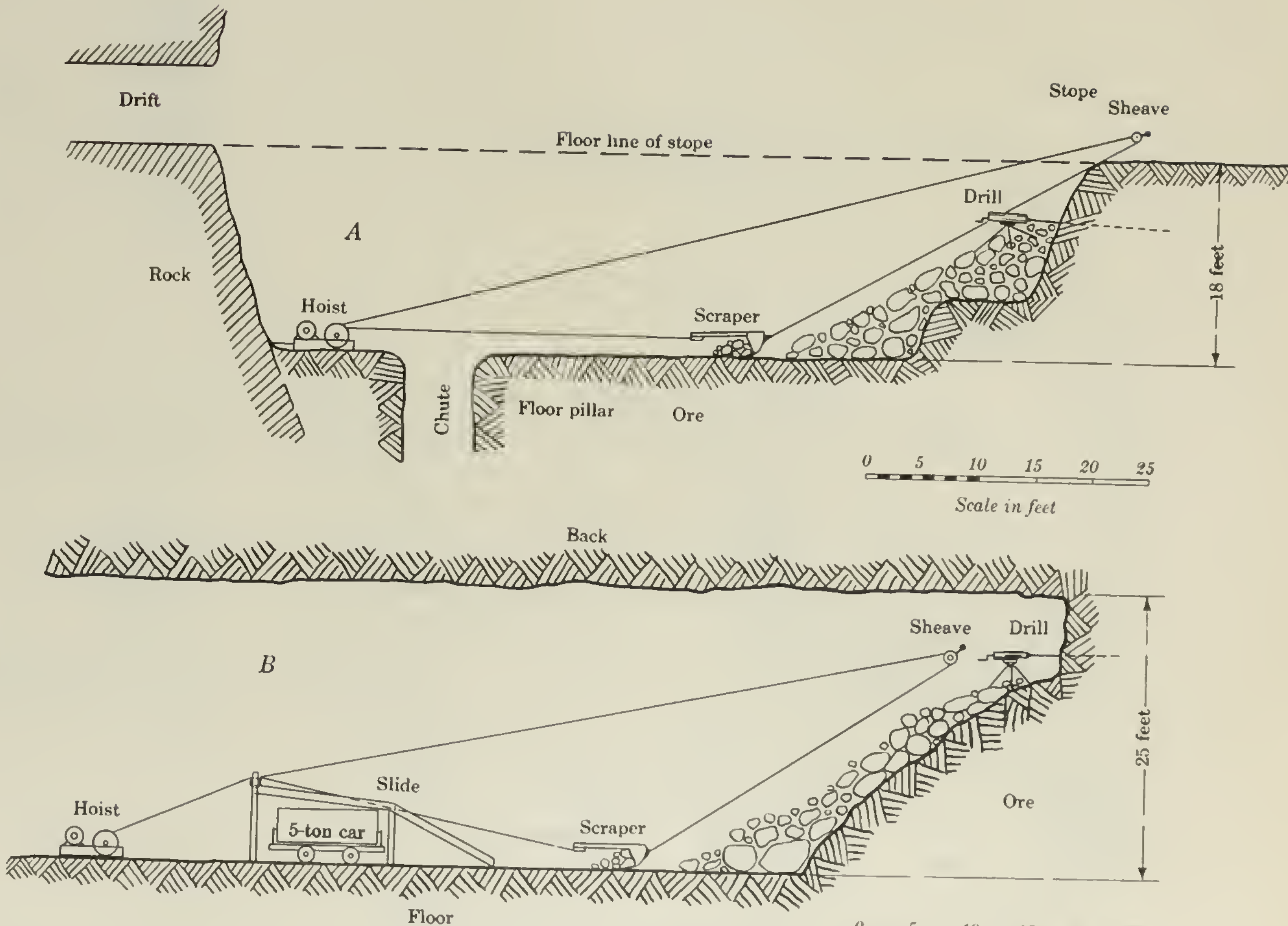

Figure 6.— A, Method of removing floor pillars.
B, Method of breast stoping

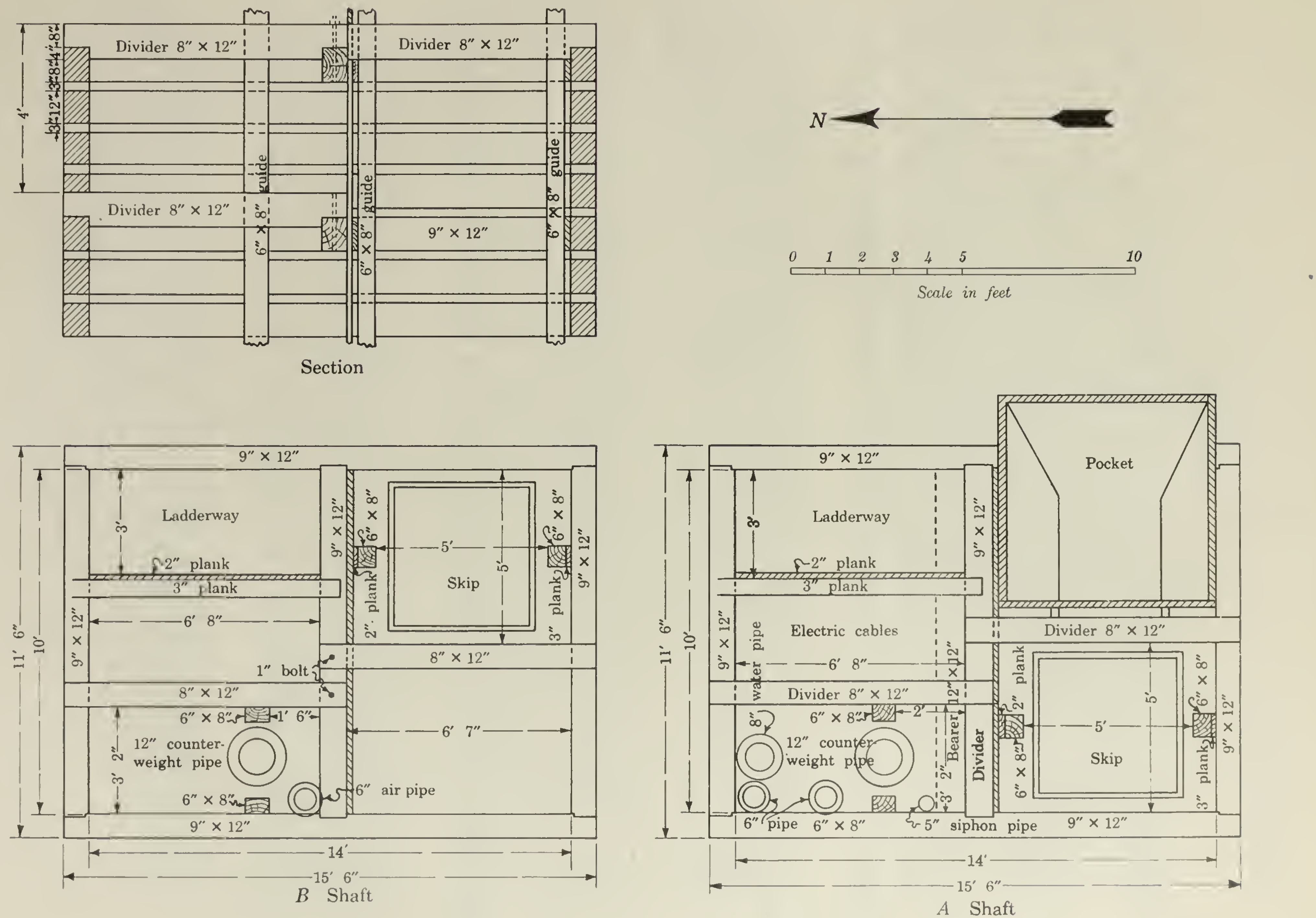

Figure 7. – Shaft arrangement

DEVELOPMENT

Shafts.- The mine is opened by two shafts, "A" and "B", 820 feet apart. The shafts, which are vertical, are 1,060 feet deep, and their bottom levels are 1,000 feet from surface. They are similar in design, 10 by 14 feet inside of timbers, and are close-timbered for 750 feet. In the last 310 feet, sets were placed on 5-foot centers in the usual way, but this was found to be a mistake, as the ore that was spilled from the skips cut the wall plates away, and inside lagging had to be put in. There are 15 levels. The first level is at 340 feet from surface; the intervals between the next four levels are 40 feet each, but below the fifth level the interval is 50 feet. The levels between the tenth and fifteenth, however, are not connected directly with the shafts, and all the ore below the tenth level is transferred to the fifteenth level by means of chutes.

Figure 3 and figure 4 show the outline of the ore bodies and the development on two levels of the mine.

The shafts originally had two compartments, each 6 feet 8 inches by 10 feet. One compartment was for the cage and the other for the Cornish pump, pipes, and ladders. In 1910 the arrangement was changed by substituting a 5-ton skip for the cage and by adding a counterweight. The skip road was moved to one end of the cage compartment, and a pocket was built at the other end on every active level (fig. 7).

On account of the hardness of the ore and the large chunks handled, the skip is very heavily constructed and weighs approximately 10,000 pounds. It is provided with safety catches, and has a small platform on top of the bail, and has a bonnet on the rope for the protection of men when riding. The skip is overbalanced by a $6\frac{1}{2}$-ton counterweight which travels inside a 12-inch pipe in the ladder and pipe compartment of the shaft. The counterweight consists of cast-iron cylinders, $11\frac{1}{2}$ inches in diameter and 5 feet long, strung on a 3-inch wrought-iron eyebolt 31 feet long. It is free to turn in the pipe.

Drifts and Crosscuts.- Except in the slate and diorite the ground in the mine is very hard, and nowhere is timbering necessary for the support of drifts and crosscuts. Practically the only timber used in the mine is that in the shafts, chutes, and ladder roads. The older drifts were driven 7 feet high and 7 feet wide, but the standard drift is now 8 feet high and 8 feet wide. On main levels where trolley locomotives are used wider spaces are excavated every 100 feet for safety zones.

The usual round drilled has the cut above the center, and two cuts are taken out before the bench is blasted out. This is gradually being changed to a center-cut round, which is squared up at the same time that the cut is blasted. In drifting, 60 per cent gelatin dynamite in $1\frac{1}{2}$ by 8 inch sticks is used, and the holes are fired by caps and fuses.

A typical round is shown in Figure 5A.

The numbers show the order in which the holes are fired.

The drilling and blasting are done on the day shift, and the broken rock is loaded and hoisted at night. All waste rock is dumped underground and is used as filling.

Drills of the Water-Leyner type, mounted on 3-inch single-screw bars $7\frac{1}{2}$ feet long, are used entirely in drifting. In very hard drifts, or in those in which rapid progress is desired, two machines are used on one bar. In other drifts one machine only is used. There is one miner for each machine.

The new drills used are of several makes and the latest models; all are $3\frac{1}{2}$-inch machines weighing approximately 150 pounds. All machines use anvil-block chucks and $1\frac{1}{4}$-inch hexagon hollow drill-steel. Double taper cross bits with 105° cutting edges are used, the gauge changing three-thirty-seconds inch for every foot of length. Sizes are arranged so that a 10-foot steel will drill a 1 5/8-inch hole.

Compressed air is delivered at a minimum pressure of 75 pounds per square inch at the drill, and water at 50 to 75 pounds per square inch. The average air consumption is 75 cubic feet per minute per machine, including line loss, at peak load.

Loading is done nearly all by hand. A dipper-type loader has been used successfully, but as it requires that the air-compressor be run at night it is not economical, because loading and hoisting of rock is the only work carried on at night. Experiments with a scraper-loader mounted on wheels and driven by electricity have been very successful and more of these machines will be used in future; loading was done at the rate of 20 tons an hour.

Drifting is done by contract, the miners being paid by the foot. They are charged for explosives, carbide, and hand tools, but drill machines and accessories, steel and compressed air are furnished without charge. Shoveling and tramming the rock is charged against the cost of drifting, but is not included in the contract.

In softer ground it is possible to drill and blast a round once a shift, but in hard ground this is accomplished only twice or three times a week.

Raises.- Raises are 6 by 6 feet in cross section. They are usually driven at an inclination of 45 to 55°, but the majority are inclined at the steeper angle. A short pyramidal cut similar to that used in drifting is generally used. In hard ground a $3\frac{1}{2}$-inch drifting machine mounted on a short 3-inch bar, the same used in drifting, is employed; but in the softer ground a dry stoper, using seven-eighths-inch quarter octagon solid steel is preferred.

No timber is used in the raises, and the miners work standing on stages made of 2-inch plank resting on drill-steel. Chain ladders, made of three-fourths-inch round iron, hung from iron pins driven into short drill holes are used for the men to climb on. These are left in the raise during blasting, and are often bent and sometimes broken, but in general they serve their purpose well.

Stoping

Mining is done by means of open stopes with pillar support. A large percentage of the floors between the pillars is recovered, but no attempt is made to recover the pillars themselves, as the ore is overlain by quicksand and much of the mine under the city.

Where the thickness of the ore will permit, rooms 25 feet wide and 25 feet high are driven by breast stoping, and floors 25 feet thick are left between levels. On the upper levels the floors are only 15 feet thick.

The rooms are advanced by breast stoping (figs. 5B and 6B). A cut 7 or 8 feet high and as wide as the stope is driven 10 feet ahead of the bench, which is taken up afterward by lifting holes drilled horizontally. The cut is driven by slabbing holes, and advantage is taken of slips wherever possible. The success of the method depends largely on keeping the corners ahead and properly squared up. When the cut is far enough ahead, the breast is drilled again but not blasted, and the broken ore is cleaned off the bench. Lifting holes 8 to 10 feet deep are drilled 5 or 6 feet apart in horizontal rows with a burden of approximately 5 feet. The whole breast is then blasted at one time. As the bottom holes do not reach in as far as those above, a toe is left as a start for a new bench, and the broken ore lies on this toe in a pile high enough to allow the miner to reach the back at the breast conveniently. The back and breast are then trimmed, and drilling for the next cut is started. This method has many advantages. It facilitates trimming the back, always leaves a bench or a pile of broken ore for the miners to stand on, and provides a fairly continuous supply of broken ore for the trammers. It also gives a good opportunity for picking out rock before it has been discolored by the ore. As there are many seams of slate in the ore: this is very important.

When the ore in a breast stope does not extend up to the next level, it must be mined by back-stoping. In starting this operation a stage is built with ladders and 3-inch planks, and from it as many holes are drilled as possible. These holes are charged, the stage is taken down, and the holes are fired. The stage is erected again, more holes are drilled, and the operation is repeated until the pile of broken ore is high enough for the miners to reach the back without the stage. The stope is carried up to the top of the ore, and is then advanced longitudinally over the stope below. The ore is broken down by horizontal holes drilled in rows; these holes can often carry a 6-foot burden. At this stage the trammers usually start at the other end of the pile of broken ore, and tram it out as the stope advances, always leaving enough ore for the miners to stand on.

<u>Extracting Floor Pillars</u>.- In mining floors, present practice is to work on a sublevel instead of mining the entire 25 feet and dropping the ore to the level below (fig. 6A). In starting, a raise is put up, a chute erected, and the ore around the raise is milled into it by underhand work down to a depth of 18 feet. When the ore will no longer run into the chute by gravity, it is shoveled by hand and later dragged by scrapers. The scrapers are particularly effective in this work because the footwall is often so flat that the ore would

have to be rehandled several times if it were shoveled by hand. Raises are usually about 100 feet apart, as the scrapers can easily reach 50 feet in either direction from the chute. When the ore has all been mined down to the floor of the sublevel, the remaining 7 feet can be mined in one lift and dropped to the level below.

Breast stopes (fig. 5B) are advanced by driving a 7-foot cut along the top of the breast for the full width of the stope. The holes are 7 to 8 feet deep, for which 50 per cent extra-low-freezing ammonia dynamite constitutes the charge. When a heading has been sufficiently advanced, it and the lower bench are carried ahead by simultaneous blasts. The same kind of machines and steel are used for drilling as in drifting, but the machine is mounted on a light tripod without weights instead of on a bar.

About five hours of drilling is done per shift; the machines drill from 5 to 6 feet per hour and average about 25 to 30 feet per shift. In full-sized stopes the burden is approximately 1 ton per foot of hole, and is less in the breast holes and more in the bench holes.

In the majority of cases one miner works in a stope, but occasionally there are two miners using two machines. The miners are paid by contract according to the number of cars (2.5 tons) trammed from the stope; the size of the stope, the hardness and toughness of the ore, etc., are taken into account in setting the price. Contracts are charged with all explosives, carbide, picks, shovels, and other hand tools, but are not charged for compressed air, or for drills and drill steel.

The ore breaks in large pieces, so that a good deal of sledging and secondary blasting is required. The largest pieces are blockholed for blasting, and the flatter pieces are bulldozed. This work is done by the miner as part of his contract.

<u>Distribution of Drill Steel</u>.- Each contract has its own steel, stamped on the shank with numerals one-half inch high, and receives each day the same steel sharpened that it sent up dulled the day before.

The sharpened steel is sorted after being tempered and is made up in bundles of 100 to 125 pounds, held together by hardwood wedges in a welded iron ring $4\frac{1}{2}$ inches in diameter. The wedges and rings are used over and over again, but one new wedge is used in each bundle, and on that wedge the contract number is written with blue chalk. The dulled steel is not sorted. Sharp steel is delivered by the cage-riders at the proper levels, and is carried to the working places by the miners. Dull steel is handled by the same men, but in the reverse order.

The average consumption of drill steel in 1928 was .105 pound per ton of ore and rock produced.

Loading Ore in Stopes

By hand.- In stopes where there is much rock to sort out, and in stopes the life of which will be short, it is better to load the ore by hand than mechanically.

The standard car is made entirely of steel, is of the end-dump type with the body rigidly fastened to the truck, and is equipped with 14-inch roller-bearing wheels. The car stands 37½ inches high, weighs 1,400 pounds empty and 7,000 pounds loaded, and carries a load of 5,600 pounds or 2½ tons; it has a capacity of 37 cubic feet. The track is 24-inch gauge and is laid with 20-pound rails.

In most places the tram is short, and the car is dumped into a chute by means of a cradle. Where the tram is long the trammers are supplied with a 1½-ton storage-battery locomotive.

The trammers (loaders or shovelers) work in pairs on contract; they are paid a certain price for filling the car and 1 cent for every 45 feet trammed. When a locomotive is provided, only the filling price is paid. The contract basis is as follows:

Price per car (2½ tons)

	Large piles	Stopes	Drifts	Chutes
Hand-filling	$0.47	$0.52	$0.57	- -
Chute filling	- -	- -	- -	$0.20
Tramming (per 45 ft.)	.01	.01	.01	.01
Minimum from chute	- -	- -	- -	.28
Minimum by hand	.55	.60	.65	- -

Note:- Prices are about 10 per cent higher for wet places. Prices for filling vary with the type of chute.

Loading by Scrapers into Chutes.- On sublevels and where a chute is within reach the broken ore is moved from the breast to the chute by scrapers (fig. 6A). These scrapers are all of one design, 48 inches wide and 7 feet long, of the hoe type, and weigh 1,500 pounds. The blade, side-plates, and draw-head are made of manganese steel. The blade is the only part that wears appreciably, and this has a life of 6,000 to 10,000 tons, averaging about 8,000 tons.

The scraper is hauled by a 25 hp. double-drum electric hoist, with planetary transmission, using five-eighths-inch extra plow-steel wire rope with independent wire-rope center (fig. 9). The motor is 25 hp., 900 r.p.m., 440 v.a.c., 3 phase, 60 cycle, of squirrel-cage type. A complete scraping outfit, exclusive of electric cable, costs $1,150 set up in place.

One miner and one scraperman constitute the crew in a stope.

The scraperman helps the miner in setting up and tearing down his machine, and the miner helps the scraperman rig up his scraper-blocks. The miner is paid by the car trammed from the chute and in most places the scraperman is paid by the day. This system is gradually being changed to a contract basis for both men.

Contract prices per car ($2\frac{1}{2}$ tons) dumped in the chute vary from 90 cents to $1.50, including both breaking and scraping.

Loading Cars with Scrapers.- In a few places semiportable steel scraper-slides are used to fill main haulage cars (fig. 6B). The scraper slides are not mounted on wheels, and must be taken apart to be moved. The hoist is not mounted on the slide, but is set up at one side of the track, and the ropes pass through blocks hung from the back or from the framework of the slide. Both hoist and scraper are of the same type as used in the stopes, previously described, and the crew in the stope is also the same - one miner and one scraperman.

Main-line cars stand 56 inches high and hold 76 cubic feet ($5\frac{1}{2}$ tons). Exclusive of switching, the time required to load a car is six minutes. Including delays and switching, four cars (22 tons) are loaded in an hour.

The number of places using this equipment is being increased.

The slide weighs 3,660 pounds and costs $360.

Underground Transportation

Trolley locomotives and large cars are used on three levels - the eighth, tenth, and fifteenth. On most of the other levels storage-battery locomotives are used with 2.5-ton end-dump cars. There are six trolley locomotives and 11 storage-battery locomotives in the mine.

The storage-battery locomotives vary in weight from $1\frac{1}{2}$ to 4 tons. They are all 24-inch gauge, and all but two are equipped with Edison A4 cells.

The trolley locomotives are all $6\frac{1}{2}$-ton, 250-volt, double-motor machines. The track is 30-inch gauge, laid with 40 pound rails.

The cars are of rocker-dump design and hold 76 cubic feet (5.5 tons) ore (fig. 10). Both locomotives and cars are equipped with automatic couplers. These cars are pulled to the shaft in trains of five or six, and the loaded cars are left at the shaft to be spotted by a tugger hoist, while the locomotive returns "inside" with a train of empties. An automatic switch turns the empties onto a siding as soon as the cars are dumped, in order to leave the main line clear for the next train of loads. No uncoupling is necessary except at the locomotive.

The cars are dumped one at a time into the shaft pocket, which has a capacity of only one car and acts merely as a chute through which the contents of the car are discharged into the skip. The loading time is 10 to 12 seconds.

No skip-tenders are employed; the cars are dumped by two skip-riders who go from level to level dumping the loaded cars that have been left on the sidings. The wages of these men is charged to tramming.

The standard chute in use underground is the Alaska-Treadwell finger chute slightly improved (fig. 8). For 20 feet above the chute the raise is put up at an angle of 45°, and is then turned to right or left or upward, so that the velocity of the ore in falling will be checked by the elbow thus formed. The chute itself is made of 3-inch hemlock plank and is lined with three-eighths-inch boiler plate. Chute fingers are made of 4 by 6-inch fir dressed on all sides to $3\frac{1}{2}$ by $5\frac{1}{2}$ inch, and are painted before being taken underground. The strap forming the hinge at the top is of three-eighths by 4-inch soft steel, and the wearing shoe at the lower end of the short member is a manganese-steel casting. The apron to catch the dribblings that pass through the fingers is made of steel. As all sizes are standardized throughout the mine, the various parts can be made up in the mine shops in quantity at considerable reduction in cost.

After a raise has been worn smooth, the time of loading at the chute is short; a $2\frac{1}{2}$-ton car can be loaded in 15 to 20 seconds and a 5-ton car in less than half a minute. The efficiency of these chutes has been a large factor in maintaining a large production.

PERCENTAGE OF EXTRACTION

One of the characteristics of the room-and-pillar method of mining is that a large percentage of the ore reserves is left in the mine as floors. While the breast stopes are advancing, floors on the line of haulage can not be mined unless provision is made to transfer the ore to lower levels. The extraction in wide bodies is approximately 65 per cent, but in narrow ore bodies with strong walls standing nearly vertical it may be as high as 90 per cent. The average for the mine has been 72 per cent.

The percentage of extraction can be worked out very closely by a checkerboard system, determining the percentage of ore mined in the stopes and pillars and also in the floors, and combining the percentages. The results check very closely with actual production.

VENTILATION

Only natural ventilation is needed. The downcast is through an adjoining idle mine, with which there is a connection from the eighth level in shaft A, and the upcast is through both the hoisting shafts. With a little care there is practically no trouble in distributing the fresh air to all parts of the mine.

FIRE HAZARDS

Although both the shafts are wet and there is very little timber in the mine, nevertheless a complete system of fire doors has been established, so that the incoming air can be instantly cut off and the flow of air between the two shafts can be instantly stopped by merely turning compressed air into the pipe line leading to the door latches. On each level at the shaft, in each headframe on the surface, and at the downcast airway there is a valve for this purpose.

First-aid and fire-helmet crews receive regular monthly training from the company's central organization and from the Bureau of Mines Mine Rescue car when it visits the district.

TABLE 1.- SUMMARY OF COSTS

Number of mine: - No. 1. Period covered: Year of 1928.

Long tons ore hoisted during period: 420,000.

Mining method: Open Stopes with pillar support.

Underground Costs Per Long Ton of Ore Hoisted

	Labor	Supervision	Compressed air, drills, and steel	Power	Explosives	Timber	Other supplies	Total
Development:								
In ore	$ .027	$ .004	$ .015	- -	$ .009	$ .003	$ .009	$ .067
In rock	.080	.009	.030	- -	.022	.003	.007	.151
Mining	.514	.028	.104	$ .006	.105	.002	.011	.770
Transportation. (underground)	.142	.007	- -	.042	- -	.005	.048	.244
General underground. expense	.075	.007	.004	.060	- -	- -	.034	.180
Total	$.838	$ .055	$ .153	$ .108	$ .136	$ .013	$ .109	$1.412

TABLE 2.- SUMMARY OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Number of mine: No. 1. Period covered: Year of 1928.

Long tons ore mined and hoisted: 420,000.

Mining method: Open rooms with pillar support.

	Development	Mining	Total
A. Labor (Man hours per long ton):			
Breaking (drilling and blasting)...	.112	.342	.454
Timbering and filling	.022	.051	.073
Mucking	.060	.310	.370
Haulage and hoisting	.017	.220	.237
General	.032	.096	.128
Supervision	.014	.037	.051
Total labor underground	.257	1.056	1.313
Average tons per man per shift....	- -	- -	6.10
Labor, percentage of total cost ..	- -	- -	65.
B. Power and Supplies:			
Explosives (lbs. per long ton).....	.196	.705	.901
(Kind and grade)	60% l.f. ammonia	50% l.f. ammonia	- -
Timber (b.m.)	.221	.224	.445
Power (kw.h. per long ton):			
Air compression	- -	4.33	- -
Hoisting	- -	2.04	- -
Pumping	- -	4.02	- -
Scrapers and lights	- -	.41	- -
Haulage and lights	- -	.76	- -
Total		11.56	
C. Percentage of Total Cost:	15.4	84.6	100.0

TABLE 2.- DETAIL OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

	Drifting	Raising
Size of excavation	8 by 8 feet	6 by 6 feet
Timbered or not	Not	Chutes only
Physical properties of rock and ore	Hard	Hard

	Total Development
A. Labor (Man Hours per Foot):	
Breaking	7.11
Timbering	1.40
Mucking	3.81
Haulage and hoisting	1.08
Supervision	.81
Total labor	14.21
Feet per man-shift of 8 hours	0.5634
B. Power and Supplies per Foot:	
Explosives (lbs. per ft.)	12.5
Timber (bd. ft.)	13.4
Total power (kw.h.)	62
C. Percentage of Total Cost:	
Labor	62%
Supplies	38%

* * * * * *

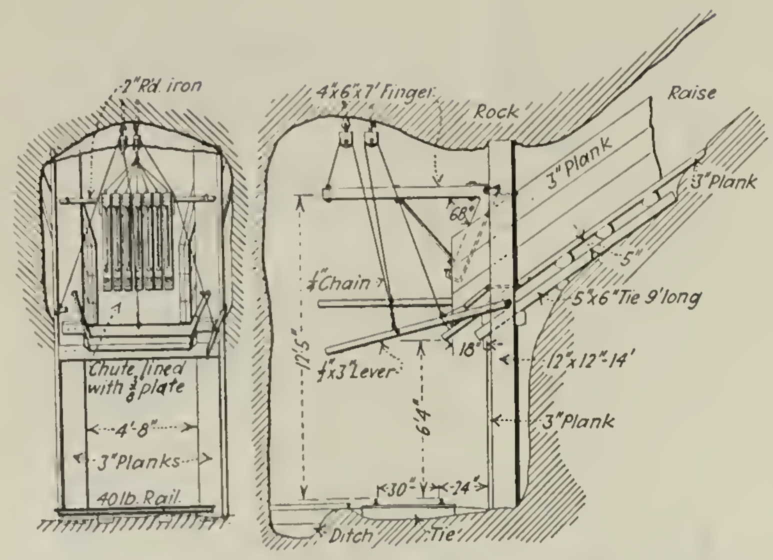

Figure 8.— Design of improved Alaska Treadwell finger chute adopted as standard

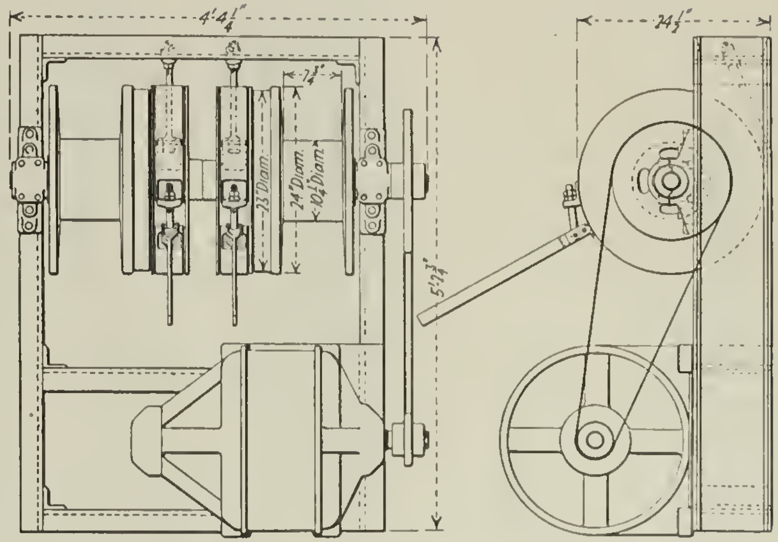

Figure 9.— Design of planetary scraper hoist

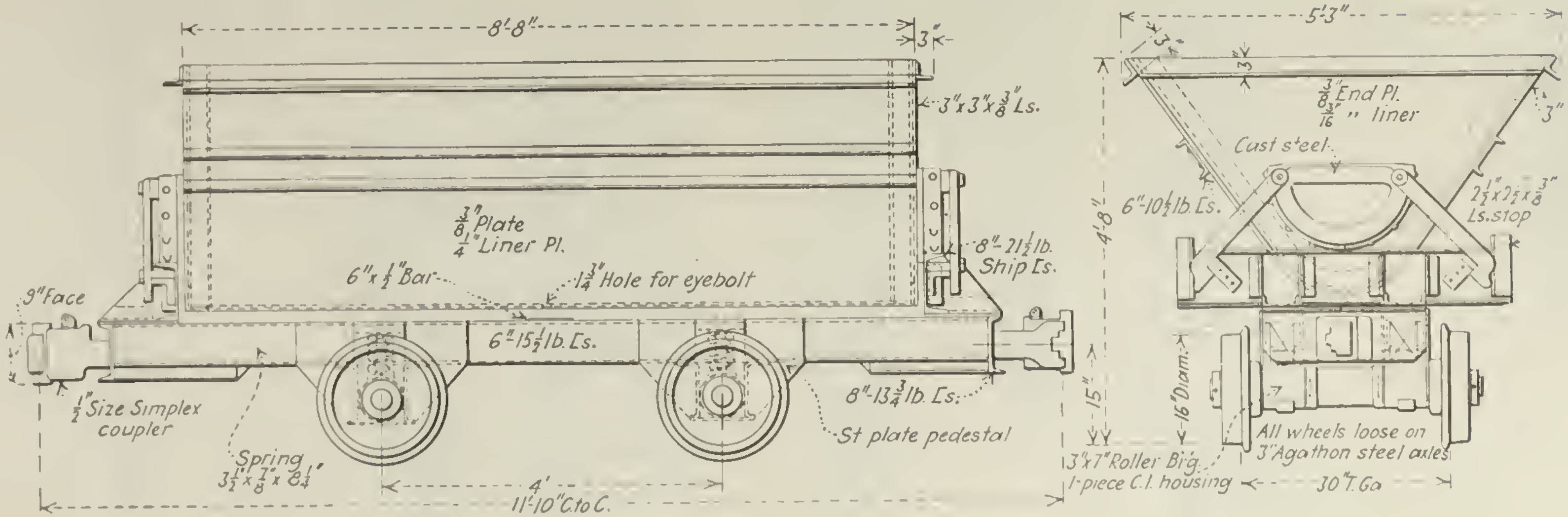

Figure 10.— Sketch of 76-cubic-feet rocker-dump ore car used on main-line tracks

Circular 6139
May, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

RECOMMENDATION FOR SAFETY IN COAL MINING RELATING TO PLACING MAIN HAULAGE IN INTAKE AIR [1]

by

THE MINE SAFETY BOARD[2]

As a basis for recommendation and for instruction of bureau engineers the considered opinion of the bureau is expressed in "decisions" of its Mine Safety Board, approved by the Director. Ten of these decisions were discussed in Information Circular 6091.[3] Decision No. 11, approved by Scott Turner, Director, on February 28, 1929, is as follows:

> "In the interest of safety, the Bureau of Mines, Department of Commerce, recommends that in coal mines, haulage and (or) hoisting be kept in intake air as far as possible."

As in the case of previous mine-safety decisions, the subject matter covered by this decision has frequently come to the attention of the Bureau of Mines through the reports of its mining engineers on explosion disasters and in accident-hazard investigations.

Although expressed in a few words, this decision is a matter of the greatest importance in the safe operation of mines, especially with the increasing use of electricity in mines. It has been frequently pointed out that since the very general use of permissible miners' lamps and permissible explosives in coal mines, the chief cause of explosions during the last few years has been electrical, and a number of these have been caused on haulage entries.

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2 G. S. Rice, chief mining engineer, chairman.
O. P. Hood, chief engineer, mechanical division.
R. R. Sayers, chief surgeon.
D. Harrington, chief engineer, safety division.
C. W. Wright, chief engineer, mining division.

3 Recommendations of the Bureau of Mines on Certain Questions of Mine Safety. Information Circular 6091, Bureau of Mines, 1928, 12 pp.

Twenty-five years ago it was comparatively rare in the generally shallow workings of that period to find a truly gassy mine, but with the increase in depth of coal mining, especially in naturally gassy fields, the number and proportion of gassy and slightly gassy mines have greatly increased.

In gassy mines it is vitally important to have the haulage and hoisting done in intake air because no one can tell when a dangerous gassy condition may arise which may lead to ignition on the haulageways if in return air, especially where trolley-locomotive haulage is in use. In nongassy mines[4] in general it is essentially immaterial as concerns hazards of explosion ignition, whether or not the haulage and hoisting are done in return air. However, as fires may occur in any mine, and in practically any portion of any mine, it is safer even in nongassy mines to have the haulage and hoisting on the intake.

Where the main haulage is on the return and a fire occurs in practically any part of the mine, the smoke and fumes from the fire quickly fill the main haulage entries, and travel of men, man trips, or man-carrying cages is made dangerous and sometimes impossible.

In many shaft mines at the time of fire, men have lost their lives or have been in imminent danger of doing so by trying to force their way through smoke and gas toward the upcast or return shaft in which they were accustomed to being hoisted; moreover, it may be the only shaft properly equipped with hoisting facilities.

It is assumed that the haulage will in all cases be done in thoroughly rock-dusted entries, as haulage by trolley or other nonpermissible locomotives is highly dangerous from the standpoint of coal-dust ignition unless the haulageways are kept thoroughly rock-dusted.[5]

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4 See Mine Safety Decision No. 3, page 4 of circular cited.
5 See Mine Safety Decision No. 5, page 7 of circular cited.

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Circular 6140 May, 1929

INFORMATION CIRCULAR

Department of Commerce -- Bureau of Mines

MINING LAWS OF BOLIVIA

BY

A. D. Garman

Circular 6140,
May, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

VIII. MINING LAWS OF BOLIVIA [1]

By A. D. Garman[2]

PREFATORY NOTE

This paper presents one of a series of digests of foreign mining legislation and court decisions which is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. This interpretation of the laws of Bolivia has been prepared from the best available information in Washington, but is released subject to correction and amplification, if necessary, by the proper American diplomatic and consular officers to whom it is being referred through the courtesy of the Department of State.

SYNOPSIS OF LAW

All useful substances of the mineral kingdom belong originally to the State. The soil and subsoil are regarded as two distinct parts, the latter extending indefinitely in depth (Art. 2). The subsoil is under the dominion of the State, which may either abandon it to common use, grant it gratuitously to the owner of the soil, or alienate it, in return for an annual fee, to any individual or association that applies for it. (Art. 1-3.)

All minerals are divided into four distinct classes, as follows:

1. Veins of any sort of metallic substances, like silver, gold, tin, platinum; metalliferous sands found on the surface of untilled lands, in the beds of rivers and running waters, and in placers; precious stones.

2. Nonmetalliferous substances, or the metalloids, such as borax, volatile alkali, iodine, alumina, sulphur, coal, nitrates, and peat.

3. The petroleums and other hydrocarbons.

1 The Bureau of Mines will welcome reprinting of this article but requests that the following footnote acknowledgment be used: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"
2 Principal translator, U. S. Bureau of Mines, Washington, D. C.

4. Building and ornamental stones; gypsum, sands, marls, emery, clays and fuller's earths, ocher and other coloring earths; pyritic, aluminous, and magnesian earths; salt-water wells and pools. (Arts. 8-12.)

Title to substances of the first and second classes may be obtained by anyone who applies for them in accordance with the provisions of the mining code. In the case of coal, however, the State holds an exemption of one-fifth in each discovery, which is not adjudicable in the ordinary form.

The exploitation of petroleums and hydrocarbons is governed by a special law, an abstract of which follows the discussion of the general laws.

The substances of the fourth class belong to the proprietor of the soil or to the public, according to local custom. But serpentines, marbles, alabasters, berengelite (asphalt), porphyry, jaspers, mines and beds of salt can be acquired by others if the soil-owner does not claim the working of them within 20 days plus ordinary traveling time after being notified of the petition of an outside party, or if he does not work them within six months after his claim is formulated. (Art. 13.)

Waste material from mines and establishments abandoned for more than six months, when located on lands without fences or walls, may be adjudicated to any one wishing to work it, on proof of such abandonment.

PROSPECTING

Any individual or association may make test pits or excavations not exceeding 10 meters in depth on uncultivated or unfenced lands, but is under obligation to indemnify the owner whenever damage results to privately owned land (Art. 15).

Prospecting is forbidden in the following cases:

1. Within the area of buildings, orchards and gardens and within towns or cemeteries.

2. Within a distance of 50 meters from public roads, canals, and railways, except with the permission of the political authority; and within the same distance of isolated buildings, unless consent of the owner is obtained.

3. On privately owned fenced lands, except with the permission of the proprietor or his representative. But in case of refusal, the prospector may apply for a special permit from the political authority of the province. If the permit is granted the owner of the land may require advance payment of damages.

The prospector has a period of 30 days (or 60, if sufficient cause is shown) after the granting of a special permit in which to begin work (Arts. 16-21).

The claim or unit of measure for mining concessions, in the case of substances of the first and second classes, is a prism with a square base of 100 meters to a side, measured horizontally, and of undefined depth. But for substances of the second and third classes this depth terminates where the exploitable material is exhausted (Art. 42).

Any person capable of legally acquiring and possessing real estate may obtain one or more claims in a well-known mineral region - that is, a region in which there is or has been at least one mine in actual operation, or as to which two or more petitions have been previously received; but the maximum number of claims for mineral substances of the first class is 30 and for those of the second and fourth classes is 64 (Arts. 22-23).

The petition (which may be in duplicate) must be presented to the Mine Superintendent of the department (or one of the departments) where the mine is situated, and must be accompanied by a map and all data pertinent to the land in question. The superintendent has power to grant an extension of time for presenting the map. The applicant must pay in advance for the required notices, which are published within 40 days.

Priority in the presentation of the petition gives preferential right.

The decree of concession or adjudication is issued on the same day the petition is presented - provided the land is found available; hence the miner can start work at once and can continue so long as papers are not allowed to lapse or legal opposition from some other miner does not arise (Title I, Chapter V).

The claim system prevails throughout Bolivia, except in a few districts like Potosi and Machacamarca, where, by reason of former concessions, which were made of the veins, the requirement of undefined depth in the case of claims can not exist. In these regions adjudications by openings (bocaminas) are practiced (Art. 41).

The petition for survey, demarcation, and giving of possession must be presented within 40 days after the end of the publication period or, in case of opposition, after the judgment has been entered. But when communication with the department capital is difficult, the superintendent has power to grant one or two 40-day extensions of time, and there is added to the first 40 days the time required for travel.

All the claims comprised in one concession must be in a single block, the contiguous claims having the full length of one of their sides in common. (Art. 48.)

The papers bearing on these proceedings are sent at once for approval to the superintendent of mines, and the applicant is obliged to urge such approval withint 30 days thereafter. (Title I, Chapter VI.)

THE MINER'S RIGHTS

The miner must state in his application the predominant mineral to be exploited. When this is any of the nonmetallic substances of the second or fourth class, he can not exploit any metallic veins or lodes, nor any petroleums or hydrocarbons, but is owner of all other veins and seams that he encounters within his claims (Art. 73). As long as he owns the mine he also owns all water encountered therein (Art. 74).

The working of mines can not be suspended or prohibited bedause of litigation, but work may be stopped in the interest of public safety, of the material preservation of the claims, or of the health or safety of the workmen (Art. 75).

Concessionnaires of minerals found on the surface may secure title to those in the subsoil by presenting a new petition (Art. 77).

The miner has the right to reduce the number of claims in his concession either at the time of the survey or subsequently, and also to abandon the whole concession (Art. 80).

Miners are obliged to arrange with the owner of the soil concerning any areas required for use in connection with their mining or metallurgical operations, and also concerning building stone, timber, fuel, or other articles needful for proper working and necessary construction. Before undertaking general prospecting, drainage, or haulage workings under nearby property, the miner must arrange with the owners of the mining rights (Arts. 82-83). If the necessary arrangements can not be made privately, recourse may be had to expropriation, on application to the subprefect; in such case the applicant must pay the appraised amount of the damage likely to result, as determined by experts. (Art. 84 ff.)

Roads built on the surface of one mining property can be used by the other miners in the same region, but the cost of upkeep of such roads is shared by all, each paying in proportion to his own use (Art. 99). Air and drainage connections, whenever the use of them involves no expense to the owner, can also be used in common, and other easements may be established subject to appropriate indemnification (Art. 104).

There is no technical surveillance as to methods of mining, but the police rules must be strictly observed (Art. 123).

"<u>Loss of Title</u>"

The right granted by the decree of adjudication lapses <u>ipso facto</u> in the following cases:

1. When the concessionnaire fails to apply, within the prescribed period, for the survey and demarcation proceedings (Art. 233).

2. When such proceedings have not been taken within the prescribed period.

3. When, in case of opposition, the petitioner does not claim the default of the opposer within the periods above indicated.

(But in all three of these cases, if the caducity has occurred through no fault of the petitioner and if the latter then prosecute his claim, the superintendent has power to remedy the defect, provided a denouncement has not already been made by a third party.)

4. When the miner renounces his concession.

The miner may also be dispossessed if he fails to pay his annual fee for two semesters.

Mining Companies

Companies formed in other countries are not permitted to operate in Bolivia until their juristic personality has been recognized there (Art. 135). This requires, among other things, a certification to the effect that at least 10 per cent of the subscribed capital has been paid up (Art. 136). Foreign stock companies whose principal interests are in Bolivia are also obliged to maintain an office and a responsible board of directors in the country (Art. 137).

Mines, like other real property, can be mortgaged; but such mortgages become extinguished by the abandonment of the mine, and unless so stipulated in the original mortgage, no personal action can be brought against the debtor (Arts. 167, 170).

Titles to mines can be transferred in the same manner as titles to other real estate, "both between the living and by reason of death." (Art. 172.)

Mines may also be leased by the owner, and the lessee is prohibited from subleasing except with the owner's permission. (Art. 178.)

Fees and Taxes

To keep his right in force the miner is obliged to pay an annual fee as follows:

1. On metalliferous veins in general, 4 bolivianos per claim.

2. On each opening in the hills of Potosi, Machacamarca, etc., 4 bolivianos.

3. On beds, strata, surficial metalliferous sands and deposits, (e.g. placers), 2 bolivianos per hectare. (Law 13 of 1880, Art. 16.)

4. On substances of class 2 (excepting sulphur, coal, and lignite, which pay the same as for petroleum), 1 boliviano per claim.

The mine engineer in the national or departmental service or the private engineer with a government diploma who is called upon for the survey and demarcation of the land and preparation of diagrams receives the following fees:

	Bolivianos
For the first claim	30
For the second claim	20
For each subsequent claim up to 30	4
From 31 to 100, each	2
From 101 to 200, each	1.50
From 201 to 500, each	1.25
From 501 to 1,000, each	1.00
From 1,000 on, each	.50

For luggage, 10 bolivianos for each day's journey.

When an accredited engineer is not abailable, any experienced man may be employed, but his fees are only two-thirds of the foregoing prices.

The authority who administers possession receives the following fees:

	Bolivianos
For the first claim	15
For the second claim	7
For each claim following, up to 30	2
From 30 on, each	1

No allowances are exacted for luggage.

There are also export duties, a tax on profits, on transfers, and on changes in organization, embraced in separate laws. The rate on transfers of mining properties is 4 per cent of the sale price (Law of November 23, 1923, Art. 1); that on reorganization (from private enterprises into general partnerships or stock companies, or from silent and general partnerships into stock companies) is 3 per cent (Ib., Art. 2).

Cancellation of Contracts

The contract may lapse or be rescinded for any of the following reasons:

1. For defrauding the fiscal interests (Art. 22).

2. For failure to drill wells in the proportion and within the periods of time specified (Art. 25).

3. For failure to pay the annual fee for two semesters (Art. 25).

4. For failure during six months to deliver the share of the gross production due the State (ibid).

5. For failure during five consecutive years to attain the minimum production (ibid.).

6. For alienating, transferring or mortgaging the concession, rights, or privileges to foreign governments, or to associations or enterprises connected with them, or admitting them as partners, or for transferring or conveying the concession to any foreign individual or corporation, without special permission of the Bolivian Government (Art. 41).

Tax on Mining Profits

(Law of November 30, 1923)

The tax falls on mining associations and enterprises generally, including those exploiting petroleum.

The rate varies between 4 per cent (on net profits not exceeding 5 per cent) and 50 per cent (on net profits of 150 per cent and over).

The rate is determined by the ratio between the net income, as shown by the balance sheet of December 31, and the actual paid-up capital, which includes the reserves not exceeding 5 per cent of the net income per year already invested in the business (Art. 1).

The capital does not include:

(a) Funds set aside for the payment of dividends.

(b) The profits obtained within the period for which the declaration is presented.

(c) Resources invested in notes or unregistered securities, such as bank stock or stock of other associations, government bonds, mortgage bonds, etc.

(d) Permanent bank deposits.

(e) Loans obtained through the issue of bonds or other forms of commercial paper.

(f) Paid-up shares issued as bonuses or honararia, whether in the form of securities or credits entered on the books.

(g) Sums withdrawn from the business and charged to the private account of the proprietor or associate. (Regulatory Decree of February 25, 1924. Art. 11.)

Net profits or taxable income is further defined as the gross receipts obtained from the production and sale of minerals, the operation of accessory industries, interest on deposits or securities in general, the rent from grocery stores, etc., after deducting expenses incident to the following:

1. Exploration and development.

2. Exploitation.

3. Ore-dressing or smelting within the country.

4. General expenses, which include:

   (a) Administration and judicial expenses.
   (b) Expenditure for hospitals, charity, labor accidents, schools and insurance on the property and buildings.
   (c) Expenses for fuel, keeping of animals, and in connection with the production of light, power, and ventilation.
   (d) Packing, transportation and storage charges.
   (e) Repairs.
   (f) All fees, taxes, etc., other than the profits tax.
   (g) Expenses in connection with accounts sales.
   (h) Interest and commissions up to 10 per cent on borrowed capital. (Ib., Arts. 19-20.)

Authorized annual allowances are the following:

1. On mining properties, buildings, and other immovables comprised in the mining business, 5 per cent on the valuation shown in the balance sheet of December 31, 1922.

2. On machinery, 10 per cent on the valuation shown in the balance sheet of December 31, 1922, or on the original value in case of acquisition subsequent to that date.

3. On animals, vehicles, furniture, and office fixtures, the actual amount of depreciation during the year, not exceeding 5 per cent of the original value. (Law of November 30, 1923. Art. 3.)

The amount of salaries, bonuses, and commissions paid to employees may not exceed 20 per cent of the production cost of the mineral or product at the place of operations. The production cost includes expenses of exploration and development, exploitation, ore-dressing, and general expenses, but does not include freight, transportation, tolls, smelting, export duties, etc. (Reg. Decree of February 25, 1924. Art. 25).

Export Taxes

Export duties in general are established for each mineral commodity on a sliding scale based upon the price of the ore or metal obtained therefrom in London. The taxes for its principal items are listed below.

On the exportation of ores of tin, silver, bismuth, and tungsten proceeding from the Departments of Potosi and Oruro, an additional tax of 40 centavos per metric quintal is levied. (Law of January 10, 1919.)

On the exportation and importation of ores from the Department of La Paz, there is an additional tax of 20 centavos per metric quintal, except for copper (q. v.). Exports to or imports from the Departments are exempt. (Law of January 14, 1919.)

There is also an ad valorem duty of one per mil on products in general imported or exported. (Law of January 22, 1910.)

Tin, Silver, and Bismuth

Export duties on tin[1]

London quotation per ton standard tin	Tax per metric quintal of 100 per cent bars, bolivianos	Add for each pound sterling, boliviano
£100 or less	3.25	
£100 - 200		0.10
£200 - 300	13.25+	0.15
£300 - on	28.25+	0.20

1 Law of January 16, 1924.

Export duties on silver[1]

Price per troy ounce, pence	Tax per Kilo of pure silver, bolivianos	Add for each penny, boliviano
20	0.40	
20 - 30	...	0.05
30 - 40	0.90+	0.06
40 - 50	1.50+	0.08
50 - 60	2.30+	0.10
60 on	3.30+	0.02

1 Law of January 12, 1924.

Export duties on bismuth[1]

(1) Price per pound of pure bismuth	Tax per metric quintal of bars or ingots, bolivianos
6 shillings	18.
6 s. 1 d. to 6 s. 6 d.	18.5
6 s. 7 d. to 7 s.	19.
7 s. 1 d. on	20.

(2) For barillas and other concentrates, 50 per cent of (1).

(3) For products of smelting - mattes, slags, etc. - whether of bismuth or copper - 40 per cent of (1).

(4) For unconcentrated or undressed bismuth ores, 25 per cent of (1).

1 Law of January 17, 1914.

Copper

Duties on copper are proportional to the grade of the ore and the cash quotation on the ton of standard copper in London, provided such quotation is not less than £60.

Export taxes on copper ores

Price, Pounds sterling	Tax per metric quintal of pure copper, bolivianos	Add for each pound sterling or fraction, boliviano
For copper ores and products of a grade not exceeding 80 per cent:		
60	0.80	
above 60		0.05
For copper or copper ore with a higher grade than 80 per cent:		
60	1.20	
above 60		0.05

The additional tax prescribed by the law of January 14, 1919, on ore exports to and imports from the Department of La Paz, is determined as regards copper as follows:

Quotation per ton, pounds sterling	Tax per metric quintal, boliviano
Under £70	No tax
70 to 75	0.10
75 on	0.20

Ores in bulk, of a grade under 20 per cent, are not taxed. (Law of February 27, 1926.)

Antimony

Export taxes on antimony ores[1]

Price per unit, shillings	Tax per metric quintal of pure antimony, boliviano	Add for each penny, boliviano
6	0.44	
6 to 8		0.005
8 on	0.56+	0.01

1 Law of February 28, 1924.

When the antimony ore contains silver or any other metal in a proportion exceeding $1\frac{1}{2}$ kilos per metric ton, the export duty on such metal has to be paid also.

Lead

The export taxes on lead ores (Law of January 12, 1924) per metric quintal is 0.20 boliviano on ingots or ores, and 0.10 boliviano on waste products (excorias). If any of these contain more than two kilos of fine silver per metric tin, the export duty on silver has to be paid.

Tungsten

Price per unit	Tax per metric quintal
Ŀ1	Bs. 1.00

For each shilling above Ŀ1 in the quotation, add Bs. 0.20 to the tax. (Law of November 21, 1918.)

Gold

An export duty of 20 centavos per ounce, on dust, nuggets, or ingots is levied on gold. (Supreme Decree of September 15, 1921.)

Zinc

An exclusive export duty of five shillings per ton of zinc ores, when the London quotation is not less than Ŀ32 per ton, is exacted. If the quotation is more, the tax is increased by six shillings for each pound sterling in excess of Ŀ32.

When the zinc ores contain silver or lead ores in a proportion not less than 20 marks per cajon (equal to 50 quintals), the duty on these ores must also be paid. (Law of January 24, 1927.)

PETROLEUM LAW

All petroleum deposits and other hydrocarbons are owned by the State, and can be explored and exploited only by the executive power, either directly or by concessions in association (Art. 1).

The distinction between soil and subsoil is maintained (Art. 2).

On enclosed and privately owned lands, exploration and drilling can not be done except by agreement with the owner and with the permission of the Ministry of Industry, with indemnification and sufficient guarantee for damage from accidents of all sorts (Art. 3).

The unit of measurement is the claim, with an area of 10,000 square meters (1 hectare). All concessions must be located without break of continuity. Concessions of more than 10 hectares must be in the form of a rectangle, with sides in a proportion not exceeding five to one.

The maximum period of an exploitation concession is 55 years, and the maximum area 100,000 hectares. The State's minimum share is 11 per cent of the gross production (Art. 5).

The executive may grant all or part of the area solicited or may deny the petition altogether (Art. 5), but only in accordance with the recommendation of the General Direction of Mines and Petroleum. (Supreme Decree of October 8, 1927, Art. 1.)

At the termination of the contract the whole equipment passes into the ownership of the State, without obligation or indemnity of any sort. If the State cares to lease the concession again, it may do so, and, other things being equal, the former concessionnaire has preference over other bidders (Art. 21). (The law fails to state whether or not in this case the successful bidder secures title to the equipment.)

Any dispute between the Government and the concessionnaire is submitted directly and without appeal to the Supreme Court of Bolivia. The concessionnaire must have in the capital a representative with ample powers for deciding any matter connected with the association (Art. 23).

Exploration Concessions

Before the concession for exploitation is granted, permits for exploration may be obtained, with the express condition that the State shall reserve one-fifth of the lands selected by the explorer as a fiscal reserve (Art. 6).

The lands applied for must be clearly determined in the petition by means of fixed starting and reference points. The maximum exploration area is 300,000 hectares. The period for making the exploration is fixed by the Government, but can not exceed four years. The concessionnaire pays an annual fee of $2\frac{1}{2}$ centavos per hectare, and deposits in advance, as a guarantee, 10 centavos for each claim (Art. 6).

Concessionnaires are preferred for exploitation concessions over the regions explored, provided they solicit such concessions within six months after the expiration of the permit (Art. 7).

Exploration permit may lapse due to:

1. Expiration of the period for which the permit was granted.

2. Failure to pay the fee of $2\frac{1}{2}$ centavos, during a year.

3. Complete cessation of exploration work for one-half the term of the contract.

4. The concessionnaire's own request - due to discouraging results of his labors.

In the fourth case, the deposit is refunded; otherwise it is retained by the State.

Concessions in Association

Proposals (whether for exploration or exploitation) are submitted to the Ministry of Industry, which either accepts or rejects them (Art. 9).

The concessionnaire (or adjudicataire) must begin exploitation work within four years after the signing of the contract; within five years he must drill one well for each 50,000 hectares; and within the next eight years he must drill a well for each 10,000 hectares in his concession. All these wells must be at least 500 meters deep, unless petroleum (or similar substance) is struck before attaining that depth.

In concessions of less than 50,000 hectares, the proportion of wells is fixed by the Ministry (Art. 13).

The location of the wells and equipment necessary for their drilling must first be approved by the Ministry (Art. 14).

The concessionnaire must furnish all the capital required for:

1. Drilling of wells.

2. Building roads for the transport of machinery and materials to the wells.

3. Constructing and operating reception plants, dams, and all sorts of storage arrangements at the wells.

4. Installing pumping equipment.

5. Constructing and operating at least one refinery in the country (if the capacity of the enterprise so requires.)

6. Constructing storage tanks for the crude petroleum and for the products obtained therefrom.

As a guarantee of the fulfillment of his obligations, the concessionaire must deposit in the Bolivian National Bank to the order of the National Treasurer the sum of 250 bolivianos for each thousand hectares, at the time of signing the contract. This deposit may be withdrawn if and when the requirements as to drilling are complied with; otherwise, it becomes the property of the State (Art. 18).

Rights of Concessionaires

Concessionaires in association with the State have the following rights:

(A) To produce, transport, refine, and sell.

(b) To install and operate any sort of equipment connected with the industry.

(c) To occupy, free of charge with the permission of the Government, the surface of fiscal lands necessary for the exploitation and deposit of the petroleums in any region within the Republic.

(d) To expropriate or burden with servitudes, pursuant to law, the adjoining lands required for the industry, by paying the amount of the indemnifications.

(e) To construct, acquire, possess, and operate, for their own use, telegraph or wireless lines, on authorization of the Government, such lines remaining subject to existing laws and regulations, and the State having power freely to make use thereof.

(f) To construct, acquire, possess, and operate railway lines, tramways, canals, roads, cableways, wharfs, in accordance with existing laws etc., with a right to the free use of a strip 20 meters wide on fiscal lands as a safety zone for railways, oil ducts, and canals; and to occupy the lands necessary for stations and their outbuildings, respecting, however, all privileges and concessions in force at the time.

(g) To import, exempt from customs duties, the equipment necessary for the exploitation work.

(h) To enjoy a privilege of a zone of 5 meters on each side of the center of their oil ducts, whether constructed or under construction, the right of crossing by other oil ducts being reserved.

(i) To organize, upon authorization of the Government, one or more companies for the development of the business.

Fees and Taxes

Concessionnaires in association with the State pay an annual fee as follows:

(a) During the period of exploration, $2\frac{1}{2}$ centavos per hectare.

(b) Beginning with the period of exploitation, per hectare as follows:

Year	Centavos
First	10
Second	15
Third	20
Fourth	25
Fifth	30
Sixth	40
Seventh	50

Other expenses are:

On adjudication petitions, whether for exploration or exploitation, of from 1 to 1,000 hectares, 20 bolivianos.

For each thousand hectares, or fraction thereof, above the first, 5 bolivanos (Reg. Supreme Decree of October 25, 1922, Art. 2).

The sum required for covering the cost of publication, three times, of the petition, and the decree of concession (amount not stated).

The transfer by purchase of petroleum properties is subject to a tax of 10 per cent on the sale price. (Law of November 23, 1923, Art. 1.)

Private petroleum enterprises transformed into general partnerships or stock companies and those which change from silent or general partnerships into stock companies, also pay 10 per cent (Ib., Art. 2). (The exact basis of this tax is not defined.)

Petroleum companies and enterprises generally are subject to the general tax on mining profits (Art. 33).

Export duties on petroleum are not provided for, and the law of November 26, 1923, declares the validity of a clause in the contract for the exploitation of fiscal reserve deposits exempting such products from every tax excepting the three already mentioned - that is, the State's minimum share, the tax on profits and the fees on claims.

The minimum production, beginning with the fifth year - end of exploration period - is fixed in the contract, and the State collects its share (in the crude or refined product, or in money) whether the amount is attained or not (Art. 33).

Special Restrictions

Only such quantities of petroleum, etc., as are not required for domestic consumption may be exported (Art. 31).

The law prescribes that at least 30 per cent of the clerical force must be Bolivians, and that Bolivian laborers must be preferred.

The executive power is authorized to reserve in each petroleum region any areas it may deem advisable, in order to hold them for direct exploitation by the State, or to lease them for exploitation by companies (preferably those formed with Bolivian capital), or to keep them as a fiscal reserve (Art. 38).

All companies are obliged to have a legal domicile in the Republic (Art. 39).

No concessionaires are permitted, in any circumstances, to alienate, transfer, or mortgage their concessions, rights or privileges to foreign governments, nor to associations or enterprises connected therewith, nor to admit them as partners. Neither are they permitted to transfer or grant the concessions to any foreign individual or corporation without the special permission of the Bolivian government.

The executive also has power to grant concessions, to associations or individuals, for the establishment of oil ducts, storage stations, and refining plants (Art. 47).

Adjudicataires of metals and inorganic substances who encounter petroleum or other hydrocarbons within their claims are not entitled to explore or exploit the latter, except by complying with the requirements of the petroleums law (Art. 49). Concessionaires of petroleum, on the other hand, are not permitted to extract the metals, etc., found in their claims, except in accordance with the mining laws (Ibid).

Mining or other provisions not in conflict with the petroleum law are applicable to petroleum concessions (Art. 50).

Petitions must bear the signature of a lawyer, to insure strict observance of the law.

EXCHANGE

At par the boliviano (100 centavos) is worth $0.3650 in United States currency, but during 1928 the average exchange value of the boliviano as reported by the Federal Reserve Board was $0.3539.

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Circular 6141 June, 1929

INFORMATION CIRCULAR

Department of Commerce -- Bureau of Mines

TENTATIVE METHOD FOR MAKING RESISTIVITY MEASUREMENT OF DRILL CORES AND HAND SPECIMENS OF ROCKS AND ORES

BY

M. W. Pullen

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

TENTATIVE METHOD FOR MAKING RESISTIVITY MEASUREMENTS OF DRILL CORES AND HAND SPECIMENS OF ROCK AND ORES.[1]

By M. W. Pullen[2]

Introduction

It is recognized that the determination of electrical resistivity of the earth within a given area may furnish information concerning the location of certain mineralized strata of economic importance. Present knowledge of the electrical resistivity of the various rock formations is almost nil, and the geophysicist who is detailed to make a survey of an area without any data as to the relative resistivity of the underlying rocks is greatly handicapped. Such surveys are usually made within areas known to contain mineral deposits to aid in locating desirable points for drilling. Usually these areas have been prospected by diamond drilling, and drill cores of the various rocks and ores are usually obtainable.

Were it possible to make resistivity tests in a laboratory of the drill cores or from pieces of the rocks and ore within an area before attempting a field survey, this would indicate not only the advisability for such a survey but would aid considerably in the interpretation of the results of such a survey. To be of most value a definite method should be used for all such tests, and as yet such a method has not been established.

This paper intends to present a tentative method and the results of tests of drill cores from the Mineville magnetite district and of hand specimens of serpentine and chromite.

Method Used for Hand Specimens

A specimen of each material, having roughly the same size and irregular shape with flat nearly parallel opposite faces, was obtained. The electrodes used were puddles of mercury 3.5 inches (8.9 centimeters) in diameter. The mercury in the lower electrode was confined in a dam of wax and the specimen set down upon it.

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1 The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be used: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"
2 One of the consulting engineers, U. S. Bureau of Mines.

The upper electrode was formed in a similar manner, the wax being built upon the specimen. Irregular readings indicated surface leakage of the measuring current, so a guard ring was arranged by making a second dam outside the upper electrode dam. The guard ring helped materially, and readings were at once more consistent; that is the galvanometer used gave steady instead of irregular deflections.

The connections for the tests are shown in Figure 1. It will be noted that the galvanometer was shunted and that a known resistance was used in the tests and calibration. The method of measurement is that of direct deflections, the known resistance being substituted for the unknown and adjusted so that the galvanometer gave equal deflections on each, the shunting resistance of the galvanometer being changed if necessary. Simple calculation then gives the value of the unknown resistance in terms of the known resistance.

Resistivity Measured with Direct Current

For the first tests, on serpentine, a single dry cell was used as a source of current and the readings taken rather quickly. Upon calibration it was found that the specimen showed a resistance of 25 megohms with the top electrode positive and 13 megohms with current reversed. It was certain that an error was involved, and as a first attempt at elimination the e.m.f. was increased by using two dry cells.

Readings were taken, and time was found to be an element in the result, as shown in Table 1.

Table 1.- Effect of time on resistivity

Polarity		Resistance,	Time,	
Top	Bottom	ohms	p.m.	Remarks: Serpentine
+	-	210,000	1:36	Reading on max. deflection of galvanometer
-	+	- - -	1:36	Battery promptly reversed; galv. off scale
-	+	504,000	1:39	
-	+	754,000	1:46	
-	+	1,039,000	1:53	
-	+	2,120,000	2:40	

The specimen of chromite was next prepared, and as a matter of course time was taken into account. The same two dry cells as above were used. The switch was closed and the set-up left for nearly 23 hours, then readings were taken with the results given in Table 2.

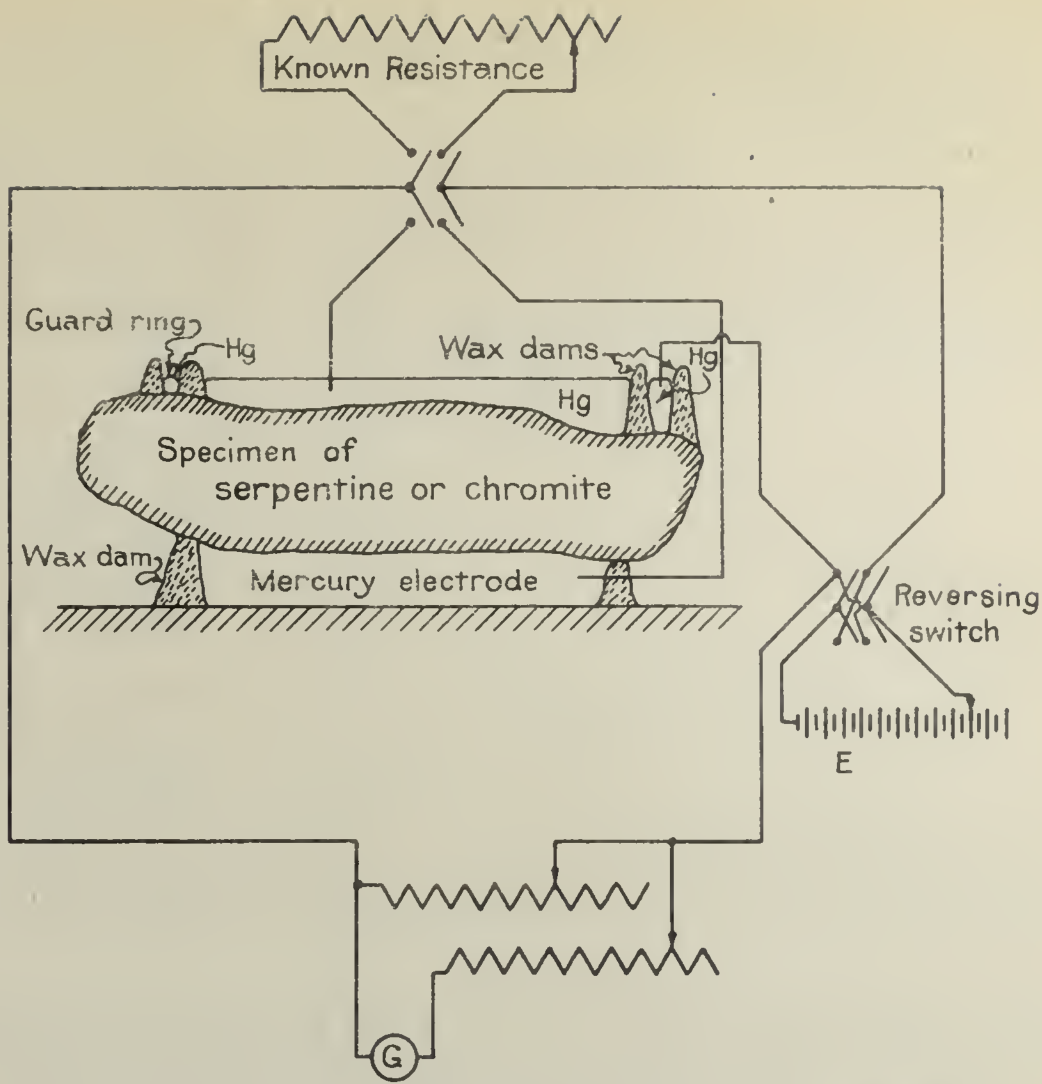

FIGURE 1.–Connections for continuous-current measurement

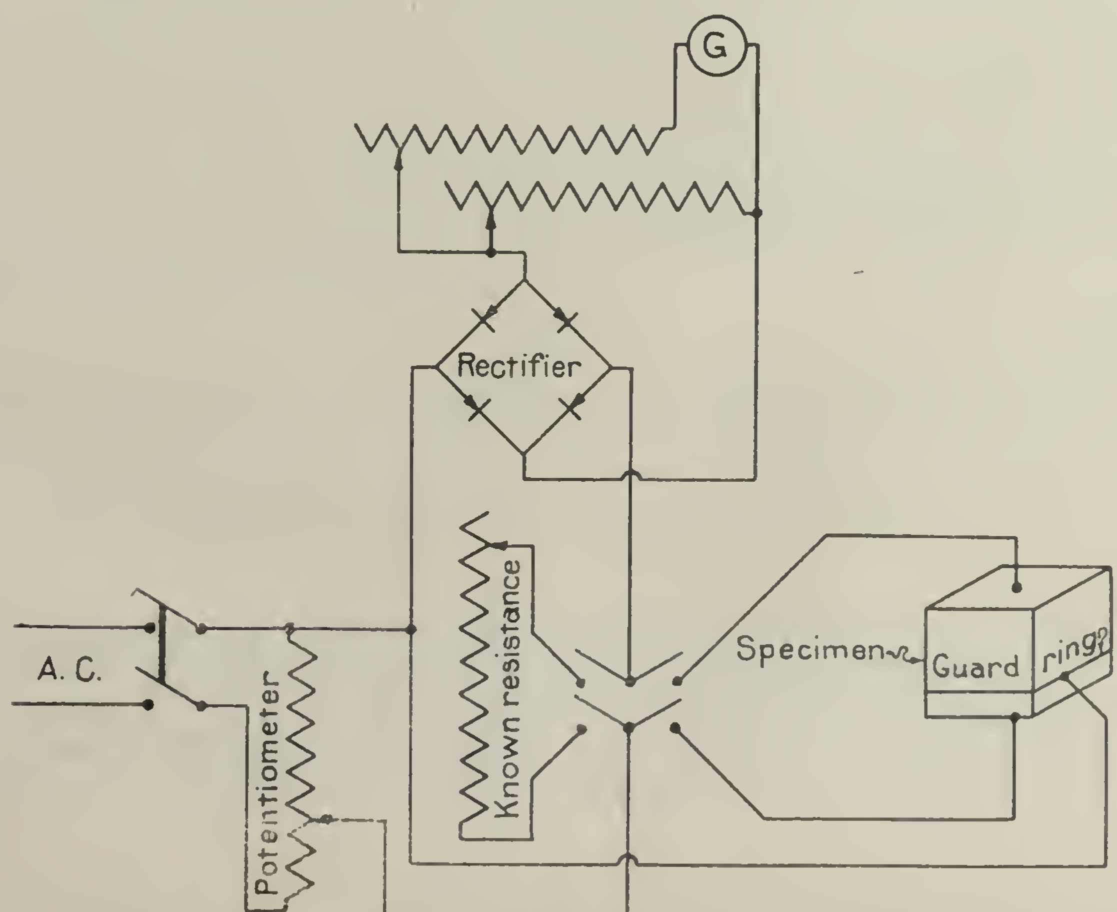

FIGURE 2.–Connections for alternating-current measurement

Table 2.- Effect of time on resistivity

Polarity		Resistance,	Time,	
Top	Bottom	ohms	p.m.	Remarks: Serpentine
+	-	7,960,000	1:54	Current on 23 hours previously.
-	+	- - -	1:55	Reverse battery; galv. deflects of scale.
-	+	3,110,000	1:56	
-	+	3,620,000	2:01	
-	+	3,980,000	2:10	
-	+	4,630,000	2:40	
-	+	5,925,000	2:42	
-	+	6,150,000	3:30	
+	-	- - -	3:31	Reverse battery; galv. deflects off scale.
+	-	4,330,000	3:35	
+	-	5,800,000	4:00	

These results also show that time is a very considerable factor in the measurements and not to be neglected, yet the proper time to use is by no means evident.

Since the two specimens above were not of such shape as to render calculation of their resistivities practical, a piece of serpentine was obtained from which a cube was cut, measuring nearly 5 inches (12.7 centimeters) on edge. For electrodes the bottom of the block was set in a pool of mercury, while a wooden frame at the top confined the mercury for the upper electrode. A guard ring was arranged by placing grooved wood strips around the block near its base. This made a very satisfactory arrangement. The connections used were those of Figure 1, and again the material showed a resistance that varied with time over several hours, as well as having a dependence on the previous polarity (see Table 1). For instance, the e.m.f. (two dry cells) was left on over night and the galvanometer deflection found to be 16.4 cm. the next morning. The battery was reversed and the galvanometer promptly deflected off scale, but gave a reading of 22.0 centimeters an hour and twenty minutes later.

A specimen of the chromite was prepared of a shape that would render calculation of its specific resistance convenient. Electrodes were also prepared. In this test wax was used to confine the mercury, as the surface of the ore was too irregular for the use of wooden strips. Substantially the same behavior was observed under the application of the electric current as in the tests of serpentine above.

In the work already outlined somewhat different results were found in the use of one and then two dry cells, so in the hope that better results might be obtained various values of voltage were applied to the serpentine specimen and the specific resistance calculated for each voltage. Time was allowed for the galvanometer to reach a substantially constant reading, as indicated in the results shown in Table 3.

Table 3.- Effect of d.c. voltage on resistivity

E.m.f. volts	Time, minutes	Specific resistance, megohm - cm. serpentine	Temp., 0^oC.
2.71	5.0	25.75	26.0
25.0	5.0	16.63	
49.5	4.0	11.15	
74.5	4.0	10.93	
98.7	5.0	10.71	
122.0	2.0	10.20	
145.0	2.0	10.18	
167.5	2.0	9.69	
190.9	5.0	9.40	27.0
214.1	5.0	9.23	
237.0	5.0	9.00	27.5
259.5	5.0	8.72	
282.0	5.0	8.34	28.0

In this table it will be noted that the initial reading was taken after five minutes electrification, and in view of the previous results the specimen could have been expected to show even higher values for a longer time. However the galvanometer motion, if any, was so slow as to be unnoticeable over a short period, and the value given is that which would be obtained by ordinary experimental procedure. In the balance of the table the readings were taken after the elapsed times as noted. No attempt was made to short-circuit the condenser made up of serpentine dielectric and mercury electrodes between readings as here given. This was tried on another occasion without any material change in the final result obtained. A very short time was required to change the value of the applied voltage, it being done by shifting connections on a bank of radio B batteries. Higher voltages than 282 were not attempted because the galvanometer tended to deflect due to leakage currents. Indeed, on a day of high humidity it was impossible to work at all with the apparatus available.

The data given indicate that the material tested showed very marked effects from dielectric absorption and polarization. The polarization shows up much more on the lower voltages than the higher values. However, a complete discussion of these effects is not possible in this paper, and for that reason the apparent specific resistances as obtained are given.

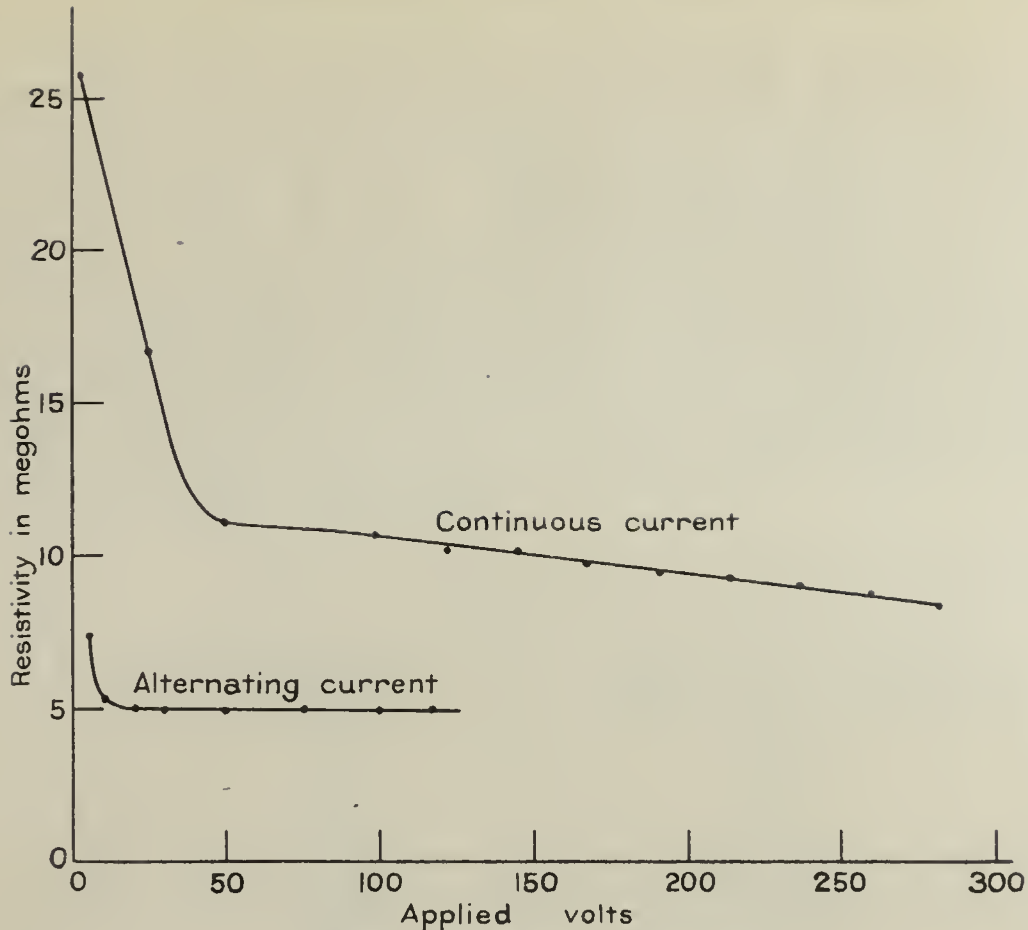

FIGURE 3.–Variation of resistivity of serpentine with applied voltage

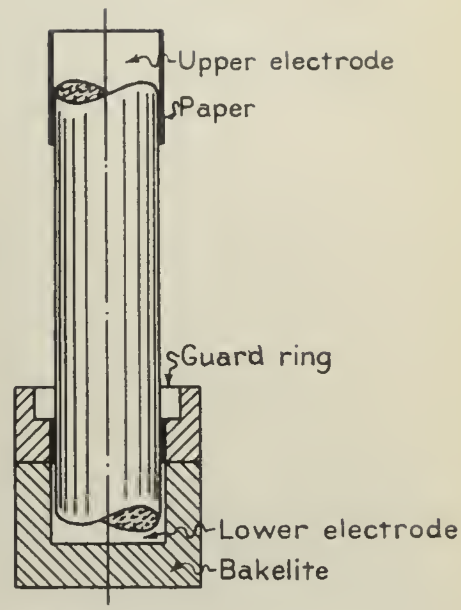

FIGURE 5.–Arrangement for mercury electrodes with diamond-drill core

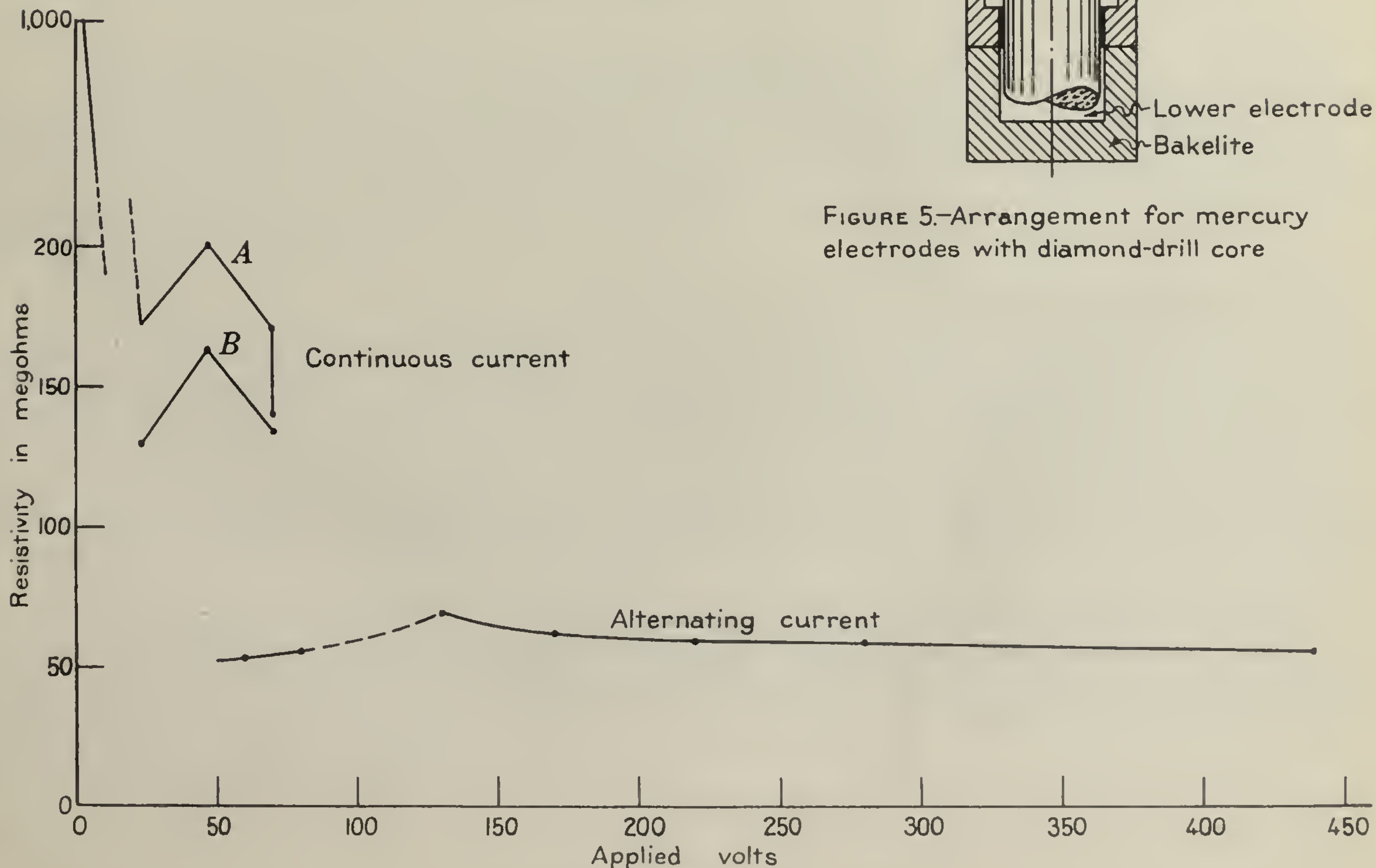

FIGURE 4.–Variation of resistivity of chromite with applied voltage

Resistivity Measured with Alternating Current

Since reversal of polarity on the specimen gave erratic readings with the continuous current, alternating current at 60 cycles was tried and the specific resistance at various voltages determined. In this case the current through the specimen was rectified with Grondahl copper oxide rectifier so that an ordinary galvanometer might be used. Connections for this test are shown schematically in Figure 2. The resistance was determined by switching from the specimen circuit to that of known resistance, which was adjusted so that the galvanometer gave the same deflection on each. A quick change-over switch was used and several checks taken at each value of voltage used. This procedure eliminates any probability of error in rectification in the rectifier, and has the advantage of giving as nearly a direct reading as can be obtained with simple apparatus. The scheme works surprisingly well, and the results obtained on the serpentine specimen are given in Table 4.

Table 4.- Effect of a.c. voltage on resistivity

E.m.f., volts	Specific resistance, megohm - cm. serpentine
5.0	7.35
10.0	5.36
20.0	5.06
30.0	4.98
50.0	4.95
75.0	4.93
100.0	4.89
117.0	4.88

Here it will be noted that with alternating current the specific resistance, as measured, is less than half the value as measured with continuous current for values above 50 volts and at lower values of voltage the difference is even greater. Further, it is noticed that the value of specific resistance as obtained becomes practically constant at a comparatively low voltage, which is not the case with continuous current.

The two curves in Figure 3, showing the relation between resistivity and voltage for the serpentine block as measured, indicate that whereas the values for continuous current have not reached a constant value for the highest voltage used those for alternating current are practically constant at and beyond 30 volts. To put the situation in another way, since the block stands 12.45 centimeters, an e.m.f. of 2.4 volts per centimeter of length gives a value of resistance which is substantially correct, whereas with continuous current a value ten times as great gives a result that is doubtful and subject to variation with time of application.

The trend of the curve for continuous current is such as to lead on to surmise that had the applied voltage been high enough the resistivity value obtained would be the same as for alternating current. At the time the measurements were made the truth of this surmise could not be tested, having the safety of the apparatus available in mind.

Measurement of the resistivity of the chromite specimen was by no means as simple as the measurement of the serpentine because of its higher value. Direct substitution of the known for the unknown resistance could not be made, for so high a resistance was not available. However the values given were checked carefully and should prove of value. Table 5 gives the data on the continuous current measurements and it will be noted that the time of application of voltage is given. The chromite shows the effect of polarization and dielectric absorption to a greater extent than serpentine, hence the values given are reliable only for the times stated and thus are subject to variation under other conditions.

Table 5.- Resistivity measurement of chromite

E.m.f., volts	Specific resistance, megohm - cm.	Temp., °C.	Remarks: Chromite
2.7	1,008.0	22.0	1 hr. 15 min; value still changing.
23.5	129.0	22.0	3 minutes
	171.6	22.0	40 minutes later
47.0	163.0	25.4	3 minutes
	201.0	25.4	6 hours later
	134.6	25.4	1 min., following value just above at 47.0 v., 6 hrs.
70.0	139.5	25.4	2 hours, 14 minutes later
	170.7	24.0	24 hours

Temperature undoubtedly has some effect, but the changes due to this cause are small indeed compared to that of time or voltage.

Alternating-current tests of the same sample of chromite as was used with the continuous current give values of specific resistance varying with applied voltage, but no worse than in the case of serpentine. Moreover the tests were difficult to make because of the very small currents involved, even at rather high voltages. The results given, Table 6, have been checked several times and are thought to be reasonably accurate. Readings below 60 volts were uncertain and have been discarded.

Table 6.- Resistivity of chromite

E.m.f., volts	Specific resistance, megohm - cm.	Temp., °C.	Remarks: Chromite
60	52.5	30.0	Connections as in Figure 2.
80	54.8	30.0	Do.
130	68.6	26.5	Transformer and potentiometer used
170	61.2	26.8	Volts read on primary and multiplied
220	58.3	27.0	By the ratio of transformation
280	58.8	27.0	Do.
440	55.2	27.5	Do.

Here again, as with serpentine, the resistivities as measured with alternating current are much lower than with continuous current. Moreover, once the set-up is made the values are more quickly determined, since the alternating current eliminates the time factor, and one has only to wait till the galvanometer comes to rest, a matter of a few seconds at most. The voltages used on the chromite do not cover exactly the same range, though there is enough overlap to give an excellent idea of what is going on. This is seen better in the curves plotted in Figure 4. In this figure the points for continuous current are erratic and there seems no reasonably short method of getting more consistant results. The difference in values of resistivity shown in points A and B are due to different times of application of the test voltage (see Table 4). The values of resistivity on alternating current are not quite consistent as between points at 60 and 80 volts and values above 130 volts. Current for the lower values of voltage was taken from the lighting mains as shown in Figure 2, whereas those for higher values were obtained by stepping up the voltage with a potential transformer run inverted and hence not in a proper working condition. The readings at the higher voltages are, however, consistent among themselves and show the same trend as in the case of serpentine tested on alternating current.

Method Used for Diamond-Drill Cores

Attention was now turned to the diamond-drill cores, of which some 140,000 feet were available for testing. The cores are of various lengths, up to over 2 feet. As a large number of measurements had to be made, two blocks of bakelite were turned up in such shape as to hold the mercury for the lower electrode and the guard ring. A paper collar confined the mercury for the upper electrode. Figure 5 shows a sectional view of the arrangement and Figure 2 the connections. The core diameter is about 7/8 inch (2.22 centimeters); and lengths up to about 7 centimeters were used, although if possible shorter lengths were preferred, as giving higher values of volts per centimeter.

From an electrical standpoint it was found that the cores fell, roughly, into three classes: (1) Those of such low resistance as to be measurable by direct comparison with a known resistance, (2) those of such high resistance that only an estimate could be made of the probable values, and (3) those in a mid-region, too high for direct comparison, yet measurable by the "method of direct deflections," a standard method in continuous current work, but here used with alternating current and the Grondahl rectifier. In this group the readings were consistent, and while it is not claimed that they are perfect the values are reasonable. The applied voltage used was 90 volts, under which the galvanometer would, on the highest resistance specimens, barely deflect. On the other hand, for the low resistances almost any desired deflection could be obtained by shunting the galvanometer. The minimum voltage gradient was thus not less than about 12 volts per centimeter.

Resistivity Measurement of Drill Cores at Various Depths

The ground resistivity of the area from which some of the cores had been taken had been measured by Dr. F. W. Lee[3] so in order to give point to further tests, measurements were made on the resistivity of the cores from the same locality. The records give the depths of geological data on the cores and were used in determining which cores should be tested.

Rather than tabulate the mass of data obtained, the results have been plotted and are shown in Figures 6, 7, and 8. In each instance the figure shows the geological formations and the depths at which they occur. Alongside is shown the curve of resistivity. The curve is not continuous, as only a few cores in each formation were tested. For values that were too great for accurate measurement the points are shown as solid circles and the curve dotted to them.

Examination of the curves shows that the values of resistivity of a given material differ among themselves. When the measurements were made it was observed that in many cases a core stated to be a certain mineral had a different resistivity from a core denominated with the same name even though from the same approximate depth. Superficial examination showed that the cores looked much alike but were not of exactly the same appearance. That they were different even though named alike is evident from the resistivity values. The "ore" mined by the company is magnetite, and this is what is meant by the term "ore" in the drill records reproduced here.

Comparison of the three curves given shows a striking similarity between them. As the ore bed is approached from above the resistivity is found to be very high, then it drops rapidly to a low value for the mineral just above the ore. The ore itself on test shows a resistivity that is higher than the rock above it unless the ore is very rich. In all cases the cores of the ore had been split for analysis, thus making measurements of resistivity difficult and uncertain. The resistivity of the ore is high, as the magnetite is segregated in discrete particles embedded in rock of high resistance. Only a very rich ore shows a low resistance, although the rock just above the ore has a low resistance. Below the ore bed the resistivity again raises to a high value, after a short distance of low-resistivity mineral.

In the curves shown there are apparently some exceptions to the observation that the ore bed has just above it a high, then a low, followed again by a high resistivity. However, comparison of the curves shows that the exceptions are only apparent but for the depths, 600 to 650 feet in Figure 7, hole No. 345. In all other cases the low resistivity persists for only a very short distance.

Effect of Moisture on Resistivity of Drill Cores

It is to be noted that the data on the diamond-drill cores were taken with the cores dry as stored in the core shed. They were laid in boxes open to the air and had been thus for several years. Consequently, it was desirable to determine the effect of moisture on the resistivity. To this end cores were selected from hole No. 348 and soaked 24 hours in mine water. The cores were then wiped off and allowed to surface dry for one hour and then their resistivity measured. Table 7 gives the results of the test.

3 Lee, F. W., Measuring the Variation of Ground Resistivity with a Megger: Tech. Paper 440, Bureau of Mines, 1928, 16 pp.

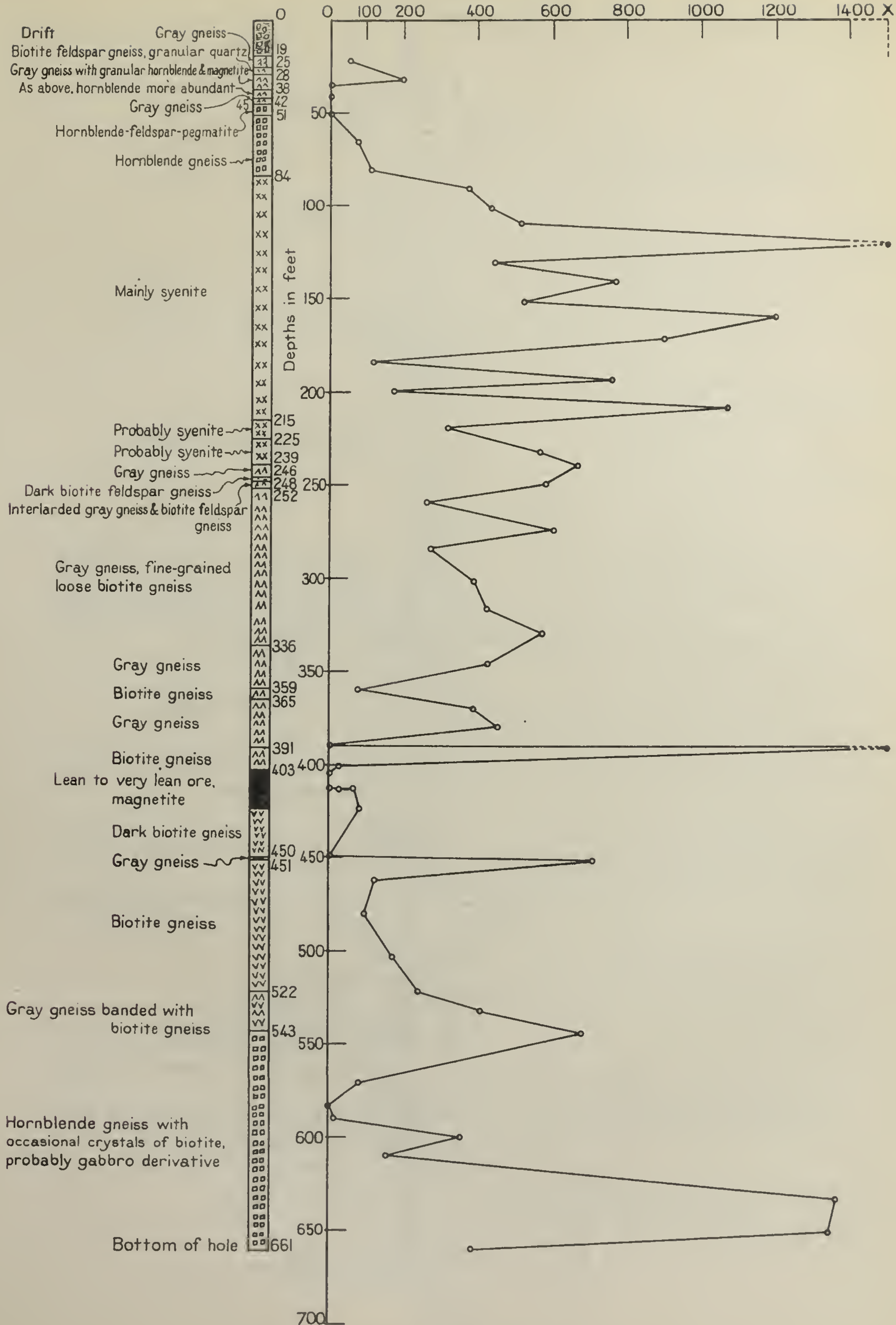

FIGURE 6.—Drill hole 338. Resistivities of diamond-drill cores for depths shown

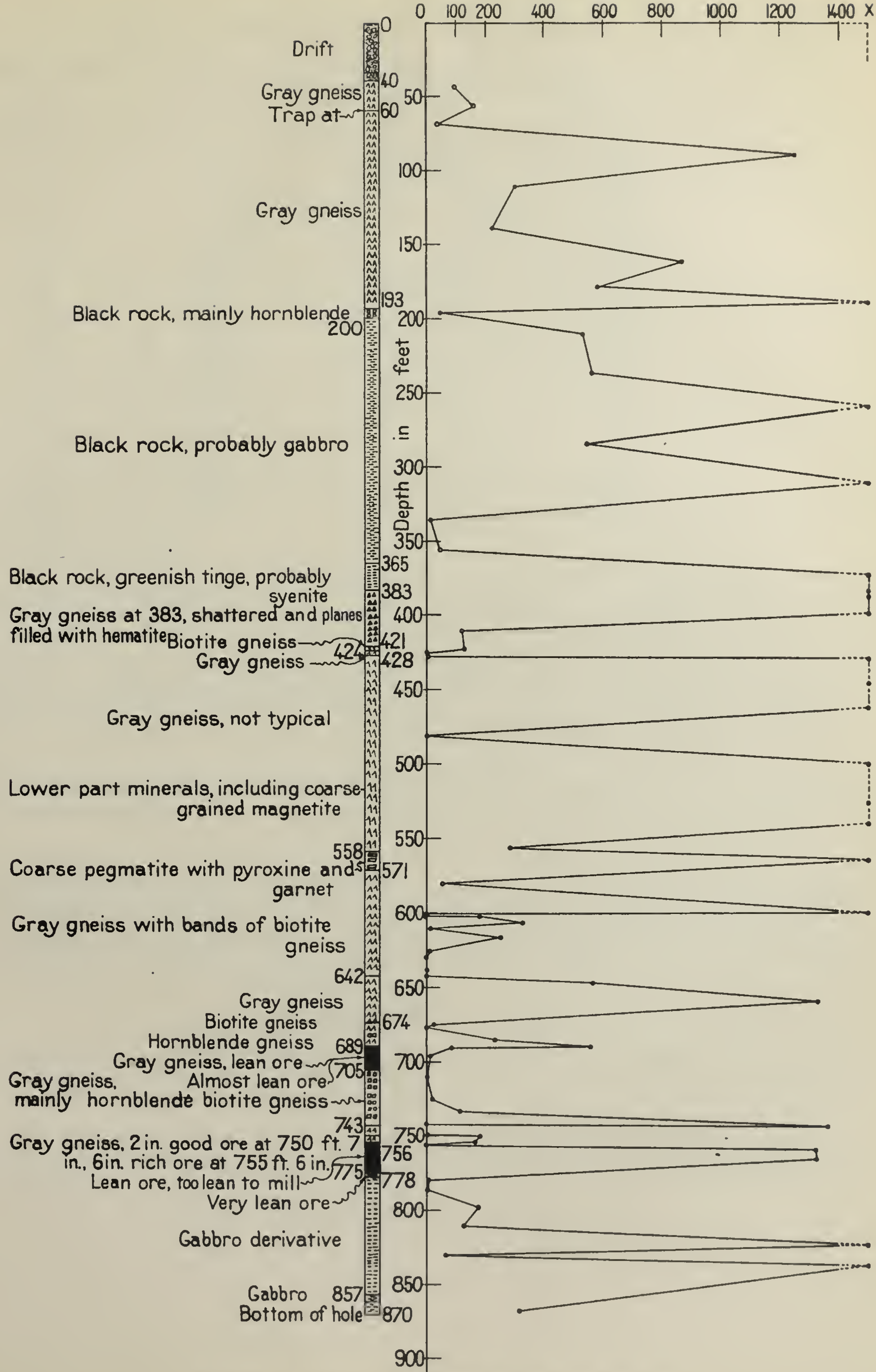

Figure 7.—Drill hole 345. Resistivities of diamond-drill cores for depths shown

Table 7.- Resistivity at various depths

Hole 348 core Test No.	Depth, feet	Resistivity, megohm - cm.	
		Dry	Wet
1	32	1/2,730	3.65
3	45	37.3	1.216
5	71½	.503	.279
10	96	.355	.325
13	112¾	47.3	1.203
20	201	1,123	342.0
23	300	1/3,480	1/2,830
29	377	1/4,075	1/2,310
33	420½	.928	.848
37	499	1/2,250	148.0
42	602½	1/3,860	91.0
46	674	3.32	.603
52	748	1.293	.772
57	815	222.5	114.2
63	915	65.75	7.78

1/ Estimated value.

In every case the soaking in mine water reduced the resistivity of the core tested. The relative change among the cores varies very widely, and apparently upsets the observations made for the dry cores. However, if the values of resistivity wet are noted on the curve of Figure 8 it will be found that these observations still hold good. The points for resistivity wet are shown in the larger circles joined by the heavy dashed lines. Continuous current methods of measuring the resistivity of minerals are uncertain in their results.

An alternating-current method of making these measurements is described. This method gives consistent results of reasonable reliability and is easily worked.

A large number of diamond-drill cores were tested for resistivity, and the result is shown graphically. From these curves it is evident that with a method of determining the variation of ground resistivity at various depths it should be possible to identify some of the geological formations at those depths.

Conclusions

1. Surface leakages of current must be reduced for reliable resistivity measurements.

2. Resistivity measurement is associated with electrical polarization, and the factor of time therefore enters as an element in its determination.

3. The rate of polarization differs in various rocks.

4. There is a variation of resistivity as a function of applied voltage and frequency.

5. Because of low values of measured current, d. c. instruments must be used in a.c. circuits when measuring a.c. resistivity.

6. There is a difference between a.c. and d.c. resistivity.

7. A definite method is proposed for preparing drill cores and hand specimens for resistivity measurements.

8. Rock specimens of the same classification from different horizons of a drill core may show a wide variation of resistivity due to the latitude in the rock classification.

9. Water content may greatly alter the resistivity of materials.

10. Resistivity of a conducting ore body may be high, if the particles of ore are disseminated.

Remarks

The main object in presenting these data is to invite discussion upon electrical resistivity measurement methods in order to establish, if possible, a uniform procedure in making resistivity tests. It is also important to take up the discussion of results in order to adopt a system of rock resistivity classification.

It is hoped that all working in this field will pool some of their information and experience and thereby avoid wasteful repetition of expenditures. Much work has been done upon resistivity measurements, but the details of measurement are generally omitted and therefore render the values unreliable. Also, much information of this kind has been transcribed from foreign investigations and is not sufficiently delineated geologically to be of much practical value.

There is some evidence that the resistivity of the samples measured is a function of the applied voltage gradient to the sample. This voltage gradient from applied potentials to the ground is of the order of millivolts and under. Because high-resistivity material was measured at a high voltage and would therefore introduce relative errors a method must be devised to obviate this difficulty. The frequencies used for resistivity measurement are of the order which allow only conduction currents to flow. High frequencies introduce displacement currents as well and would further complicate the problem.

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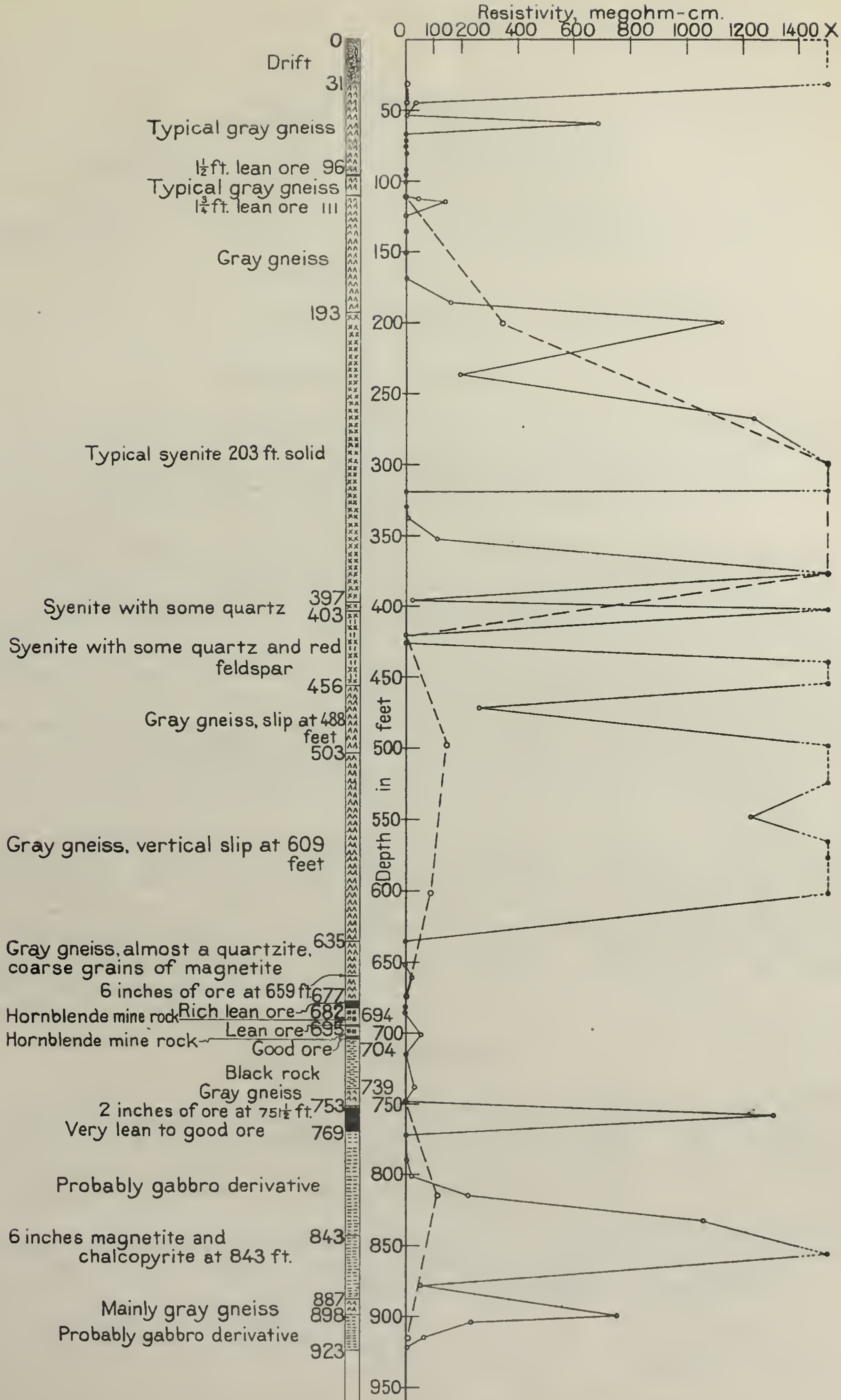

FIGURE 8.—Drill hole 348. Resistivities of diamond-drill cores for depths shown. Dashed line shows resistivities of wet cores

8-8-29

Circular No. 6142
June, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

MINERAL WOOL[1]

By J. R. Thoenen[2]

DESCRIPTION AND USES

Mineral wool is a substance composed of very fine, interlaced threads chiefly of calcium silicate or glass-like materials similar in appearance to wool or cotton. Because of the high percentage of air space for a given volume of mineral wool, it is used mainly for insulating purposes.

Insulation has two principal functions; retention (or exclusion) of heat, and sound control. Mineral wool finds a considerable market for use in high-temperature insulation as a cover for pipe, annealing and baking ovens, and metallurgical and chemical furnaces, and in low-temperature insulation as stove and fireless cooker linings. Processed in a different manner it makes an efficient insulator for such uses as filling for house walls and floors, refrigerator lining, and covering for house and underground pipe lines. Used in interior walls and like places, it is an efficient sound deadener as well. Because of its acid-proof properties it is also used as packing for acid carboys and as a filter medium for acids and corrosive liquids and gases.

Mineral wool may be divided into two classes - rock wool and slag wool. The differentiation is based on the nature of the raw materials used in its manufacture. Rock wool is made from a natural siliceous limestone (more properly classified, perhaps, as a calcareous shale) found in northern Indiana[3] or from an artificial mixture of silica and limestone. Slag wool is made from iron blast-furnace slags with or without the addition of limestone to temper the charge.

SOURCES OF INFORMATION

Numerous inquiries received by the Bureau of Mines as to the nature and use of mineral wool prompted a study of the material, and in the absence of other than very meager published literature on the subject a field trip was made to the producing plants. This paper presents the data collected during

1 - The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be used: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

2 - Mining engineer, U. S. Bureau of Mines.

3 - Cummings, E. R., and Shrock, R. R., The Geology of the Silurian Rocks of Northern Indiana. Dept. of Conservation, Division of Geology, Indianapolis, Ind., Publication No. 75, 1928.

that field study in December, 1928, and February, 1929, together with information gathered by the author during previous personal contact with the industry.

HISTORY OF THE INDUSTRY

Little data concerning the history of the industry are obtainable, although Lang[4] states that mineral wool has been made for over 50 years.

Some years ago a slag-wool plant was operated near Pasadena, Calif., but the author is informed this plant is not now in existence. Also, it is said that there is a plant in Tennessee, but no information is available concerning it.

Rock wool was probably first made at Alexandria, Ind., in 1897 by Mr. C. C. Hall, Manager of the Banner Rock Products Co. Mr. Hall at that time was employed by St. Louis interests as a chemical engineer and manager of a steel plant that was designed and erected under his supervision at Alexandria, Ind. In the search for suitable rock for fluxing purposes in this plant, the peculiar composition of a local deposit was ascertained.

Mr. Hall resigned his position and began experimenting with this rock first to see if it would melt and then to see if it would make a fiber. The abundance of natural gas suggested its use as a fuel, and it was so employed. Mr. Hall's first production of wool was on the premises of the steel plant, but as this plant was absorbed by one of the steel trusts then forming, Mr. Hall had to move such equipment as he had assembled. At this time he formed with some Alexandria friends a corporation known as the Crystal Chemical Co. This second plant was operated until about 1901 when it was sold to a St. Louis company which ultimately was succeeded by the present General Insulating & Manufacturing Co., with headquarters in St. Louis.[5] In 1906 Mr. Hall withdrew from this company and formed the Banner Rock Products Co., which was incorporated in September, 1906, and has operated continuously since January, 1907.

The Union Fibre Co., Inc., for several years operated a small plant at Yorktown, Ind., but they have recently abandoned this site and moved to Wabash, Ind., where they have built a modern plant.

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4 - Lang, Herbert, Designing and Operating a Slag Wool Plant. Chem. and Met. Eng., Aug. 27, 1923, pp. 365-367.

5 - Thoenen, J. R., Mineral Wool and Cement from a Silicified Lime Rock. Rock Products, Feb. 21, 1925, p. 39.

Mineral wool is at present manufactured by the following companies:

Companies employed in the manufacture of mineral wools

Company	Plant location	Number of furnaces	Kind of wool
Banner Rock Products Co.	Alexandria, Ind.	10	Rock.
Johns-Manville Corporation	Waukegan, Ill.	1	Slag.
Johns-Manville Corporation	Manville, N. J.	1	Slag.
General Insulating & Mfg. Co.	Alexandria, Ind.	6	Rock.
Union Fibre Co., Inc. (Winona, Minn.)	Wabash, Ind.	2	Rock.
United States Mineral Wool Co.	Netcong, N. J.	3	Rock and slag.
Columbia Mineral Wool Co.	South Milwaukee, Wis.	2	Rock and slag.
Webber Cement Insulation Products Co.	East Chicago, Ind.	1	Slag.

The Banner Rock Products Co. has recently been purchased by the Johns-Manville Corporation, and the Columbia Mineral Wool Co. is subsidiary to the U. S. Mineral Wool Co. The plant of the Webber Cement Insulation Products Co. was not visited by the author, but it is understood that this plant operates on slag from a local smelter and its whole wool product enters the market as an ingredient of insulating cements.

MINING PRACTICE

For the manufacture of slag wool, the slag is loaded from the dumps either by hand or by mechanical shovel and transported to the manufacturing plant. Some operators have located their wool plants adjacent to old abandoned dumps and do their own loading. Others buy their slag from the iron companies which then reclaim the slag from the dump and deliver it to the wool operator. Often the material is transported for several miles. So far no wool plant is known to utilize slag while still molten.

The natural Indiana wool rock, on the other hand, is mined from open quarries. Cummings and Shrock[6] give a detailed description of the geology and occurrence of this rock, and the author[7] has described the general quarry practice. The shallow depths and thin beds of the material favor the use of hammer or piston drills, but at least one company has opened a deposit of considerable thickness and plans to install churn drills.

Some of the deposits are soft enough to be mined without blasting. Elsewhere as much as 1 pound of explosive may be required per ton of rock broken.

6 - See footnote 3, p. 1.
7 - See footnote 5, p. 2.

Overburden is stripped by hand or steam shovel. The broken stone is loaded by hand or shovel and conveyed to stock piles, where it is stored during the winter season to obviate mining except in summer.

MANUFACTURING PRACTICE

Handling Materials

Both rock and slag are melted in cupolas, very much like cast iron. To convey the raw materials to the tops of the cupolas various means are employed, among which may be mentioned bucket elevators, elevator and cars, and cranes.

Coke is the usual fuel employed and is charged in alternate layers with the slag or rock in the tops of the cupolas. The charge is proportioned at some plants by counting the shovels of each material; at others monorail buckets are used, and the charge is proportioned entirely by weight.

Cupola Practice

The cupola in almost universal use is of the vertical-cylinder water-jacketed steel type, roughly 7½ feet in diameter by 16 feet in height. Blowing tuyeres are placed about two feet above the bottom, and the molten material is drawn off near the bottom through a fire-clay lined opening. Air is introduced through the tuyeres by blower fans at low pressures. The bottom of the cupola is arranged with drop doors to facilitate cleaning. The water-jacketed cupolas are said to have a capacity of 1,000 pounds of wool per hour and are operated continuously six days per week when in production.

Brick cupolas of the same general dimensions have been used in the past and are at present used by one company. The brick units operate only part of the 24 hours, and repairs to the lining are made while the cupolas are idle.

In some instances the water-jackets are used to generate steam for power purposes and for wool blowing.

Blowing Wool

The molten material issues from the bottom of the cupolas in a small stream, the flow and temperature of which are carefully regulated. The slag as it falls is broken up by a steam jet (at 80 to 100 pounds pressure) into minute balls or shot , which while still in a molten condition are propelled rapidly through the air. In passing comet-wise through the air, fine threads of glass-like material form as tails to the shot and fall in a fluffy mass on the floor of the wool room.

The best form and shape for the steam-jet blower is a subject of considerable controversy among operators. The various operators advocate their own particular design and maintain more or less secrecy concerning its details.

Generally speaking they are all so arranged as to form the issuing steam into a trough which receives the falling slag.

Gathering Raw Wool

The wool blown into the wool rooms is gathered either by hand or on conveyor belts, for further processing.

Some operators arrange two wool rooms for each cupola, so that wool may be blown into one while being gathered by hand from the other. Others blow into one room and gather the wool while the cupola is down for repairs. More modern practice involves continuous blowing into a single wool room, the floor of which is formed by a moving conveyor belt which gathers the wool automatically as it falls, and passes it on for further processing.

FINISHED PRODUCTS

Loose wool is the sole product of several plants and finds a ready market wherever a loose insulating material is required. Mineral wool is also used in loose form as a filter medium for acids and corrosive gases, as well as for packing around acid carboys for shipment.

Loose wool is run through machines termed "granulators," the function of which is to break the shot from the fine threads and remove it. This treatment results in a short-fiber, shot-free wool which is used for mixing with other materials in the manufacture of insulating cements. This material is also used extensively as house insulation. The granulated wool is blow by low air pressure into the spaces between the studding and joists of buildings already constructed and forms a very effective heat and sound insulator.

In addition to the method of using granulated wool for house insulation as already noted, raw wool is placed between wire netting such as window screening, chicken wire, or chicken wire and metal lath. These "blankets" are made in various sizes and thicknesses to suit consumers and for house insulation are placed directly on the studdings or between them. Where metal lath is used on one side, the lath is plastered in the usual manner. Similar products are made for refrigerator and cold-room linings. Blankets are also used for boiler and oven coverings, and are in turn covered by insulating cement. Covering for outdoor tanks in which liquids must be kept above freezing is a similar use for this class of fabrication.

Used in blankets or as loose wool the material is said to be of great benefit in controlling the acoustic properties of buildings, and in rendering noiseless small machinery units such as house refrigerator motors, etc.

Probably the greatest use for raw and granulated wool is in the manufacture of "rock cork" or "rock felt." Blocks of this material are made in various sizes and thicknesses for refrigerator lining. Raw or granulated wool, together with other ingredients and various binders, is mixed with water into

a stiff mud and is placed on pallets and subjected to low pressure. These pallets are then run on racks into drying rooms and dried in a current of hot air.

This material is also molded into forms for pipe covering and is covered with cotton. Often it is reinforced with chicken wire.

High-temperature insulation is made from a mixture of refractory binders with the wool; the whole mixture is compressed into blocks or brick. Such material is used for covering glass and metallurgical furnaces, annealing and enameling ovens, and for pipe covering.

Various grades of raw wool are sold, depending on the consumer's specifications. No standards have as yet been set up. Some consumers demand that the wool shall be oiled, whereas others require that it be dry. Prices depend on competition with similar insulating materials and even for similar grades are likely to vary in different contracts.

The following prices are indicative only:

Raw wool - $20 to $25 per ton.
Granulated wool - up to $40 per ton.
"Flex felt" and "blankets" - 8-10 cents per board foot.
Block - 15 cents per board foot.
Cork board - 7 cents per board foot.
Pipe covering - Standard 85 per cent magnesia list, less 50 per cent.
Brick - $100 per M.

DESCRIPTION OF PLANTS

There are many elements in the manufacture of mineral wool which must be classed as secret trade practices. Therefore the following description of plants must be confined to more or less general information to avoid disclosing private practices.

Johns-Manville Corporation

Banner Rock Products Plant, Alexandria, Ind.

The Banner Rock Products Co. operates a large open quarry of wool rock which is covered by 3 to 5 feet of clay and loam overburden. Stripping is done by steam shovel and drag line. The wool rock attains a maximum thickness of 15 to 20 feet and tapers in all directions.

The rock is drilled with steam piston and air hammer drills, blasted with 40 per cent gelatin dynamite, and loaded into small quarry cars by hand and steam shovel. Loaded cars are trammed to the foot of an incline and hoisted to a storage pile adjacent to the kilns or cupolas.

The manufacturing plant consists of the original four-cupola installation and a new modern six-cupola plant.

In the old part of the plant the cupolas are charged by hand, but in the new plant raw materials are brought to the charging floor by an overhead traveling crane which deposits the rock and coke in alternate bins near the tops of the cupolas. A traveling bucket operating on a monorail is loaded by hand from the various bins, weighed, and dumped into the hoppers over the cupolas.

The cupolas are all of the standard, vertical, steel-cylinder, water-jacketed type. Draft is furnished by blower-type fans. Owing to the hardness of the local water, it must be treated before circulation through the jackets.

In the old plant the wool is blown into alternate wool rooms from each cupola, but the rooms in the new plant are fitted with sloping sides and a narrow conveyor belt at the bottom. This insures automatic and continuous collection of wool.

Arrangements are such that the finished wool can be sent on conveyors direct to fitting tables for fabrication into blankets or to granulators and thence by elevators to the mixing tanks for the preparation of rock cork.

The whole new plant is designed to utilize all possible automatic transportation devices and thus insure constant and continuous operation with a minimum of labor.

The plant employs approximately 300 men, and has a potential capacity of 10,000 pounds of wool per hour.

Waukegan, Ill.

The equipment of the plant at Waukegan consists of one standard water-jacketed cupola and operates on slag obtained from an iron blast furnace at Mayville, Wisconsin. Composition of the slag was given as:

	Per cent
SiO_2	38.0
Fe_2O_3.	1.0
Al_2O_3.	11.0
CaO	28.0
MgO	19.0
S	0.5 - 0.8

Occasionally the charge must be tempered with other raw materials to insure the correct chemical control. The cupola is charged by hand, and the blowing practice is standard. No fabrication is attempted except in the form of blankets.

Plant capacity is 1,000 pounds of wool per hour.

Manville, N. J.

The plant at Manville, N. J., also uses slag as a raw material. Although the plant was built as a production unit, it is now used primarily as a technical experimental plant.

The raw materials are charged into the single standard cupola from overhead bins and the wool is blown into a vertical-sided room in which the floor is a conveyor belt. A short conveyor belt takes the blown wool from the collecting conveyor for sacking.

No fabrication is attempted at this plant. Plant capacity is 1,000 pounds of wool per hour.

General Insulating & Mfg. Co., Alexandria, Ind.

The plant of the General Insulating & Manufacturing Co. at Alexandria, Ind., was described by the writer in the February 21, 1925, issure of Rock Products, and the reader is referred to that article for a detailed account of operation.

The plant operates on Indiana wool rock from nearby quarries. Six water-jacketed cupolas are used, and various products are fabricated. The plant has a potential capacity of roughly 6,000 pounds of wool per hour.

United States Mineral Wool Co.

Columbia Mineral Wool Co., South Milwaukee, Wis.

The Columbia Mineral Wool Co. at South Milwaukee, Wis., likewise uses slag from the Mayville Iron Co. as raw material, but both Mayville dolomite and red granite are added with the slag in the cupola charge to obtain the correct chemical control.

The wool is blown from one standard water-jacketed cupola, using a fan-type blower for cupola draft, into alternate wool rooms and collected by hand.

No wool is fabricated at this plant.

Capacity of plant is 1,000 pounds per hour.

Netcong, N. J.

The U. S. Mineral Wool Co. operates one brick cupola at Netcong, N. J., for 16 hours daily. During the idle 8 hours the cupola is entirely relined with a local clay and the wool is removed from the blowing room. A second brick cupola and also a standard water-jacketed cupola are available but were not operating at the time of the author's visit.

At this plant both rock and slag wool are made. The slag is obtained from abandoned dumps of the Replogle Steel Co. Raw calcite is added to the slag charge to control the proper mixture.

For making rock wool, silica rock and crystalline calcite are purchased from nearby sources. The fuel charge with these raw materials consists of a mixture of anthracite coal and coke. With the slag, coke is used alone.

The blowing operation is similar to that already described, and the wool is collected by hand from the blowing chamber.

No wool is fabricated.

The plant capacity is 1,000 pounds per hour for 16 hours daily with one cupola. Operating the three cupolas, the potential capacity would be tripled.

Union Fibre Co., Inc., Winona, Minn.

Wabash, Ind.

The Union Fibre Co., Inc., recently transferred its operation from Yorktown, Ind., to Wabash, Ind., where a modern plant has been erected.

The raw material used is the Indiana wool rock, quarried near the plant. This company was particularly fortunate in its quarry site, as previous to their advent a former company had operated a commercial stone quarry on the property. This operation had removed not only the overlying clay and loam surface but also a layer of hard limestone, and as a result the wool rock was stripped and exposed over a large area. In this particular locality, moreover, the wool rock apparently attains its maximum thickness in the State. The face, as exposed in the quarry, has a vertical height of roughly 60 feet. Future plans are to operate the quarry with churn drills, carrying the face the full height of the deposit. At present, air hammer drills are used on shallow benches. The stone is blasted with 40 per cent dynamite and loaded by hand into small steel quarry cars, which are hauled up an incline and dumped on the charging floor above the two steel water-jacketed cupolas.

Cupolas are charged by hand with rock and coke, and the wool is blown by steam in the usual manner. The plans call for installing additional cupolas when the demand requires.

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Considerable study is apparent in the design of the blowing room and in the subsequent processing of the wool. The blow room itself has vertical concrete side walls 6 feet apart, and the floor is a conveyor belt which, being inclined, also acts as an elevator and carries the blown wool the full length of the blow room to the top of the factory.

Ingenious devices in the process enable the wool to be controlled uniformly for thickness and density over the full width of the conveyor. As the wool comes from the blowing chamber in a continuous blanket of predetermined thickness and density it is discharged onto a descending conveyor where circular slitting knives divide it into widths convenient for different uses and for rolling up at the lower end into rolls. These can then be wrapped in paper by suitable machinery in a manner similar to the wrapping of cotton batting.

The wool is not as yet fabricated into any product other than the blanket form, although future plans call for such fabrication as markets may require.

The potential capacity of the plant is 4,000 pounds per hour.

DISCUSSION OF TECHNICAL PHASES

General

Mineral wool is manufactured by first fusing the necessary raw materials to a liquid slag and then blowing this molten slag into wool.

The resulting wool, whether made from blast furnace slag, natural rock, or a combination of raw materials, is essentially the same for all market purposes. Technically, this statement may be disputed for the reason that blast-furnace slags often contain small amounts of sulphur which if present in the sulphide form in wool, would have a tendency to corrode any metal such as reinforcing metal, oven-sheathing, water pipes, etc., with which the wool might come in contact. However, since rock wool also is likely to contain this element, (originally in the fuel) such wool might be objected to on the same grounds.

In any event, sulphur may be considered detrimental only when it is present in the sulphide form. If it be entirely oxidized to sulphate the detrimental action is greatly reduced if not entirely eliminated. In this connection it is interesting to note the results of experiments by various investigators as reported by the National Slag Association.[8] These experiments, as listed by the association, indicate that slag wool has no corrosive action on steel inclosed within it. On the other hand, manufacturers of rock wool claim this is the fundamental difference between rock and slag wools. In the writer's opinion this matter of sulphide and sulphate content is a pertinent subject for research by wool manufacturers, as it embraces the whole question of raw materials.

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8 - National Slag Association, Is There Any Corrosive Quality in Slag. Symposium No. 10, 937 Leader Bldg., Cleveland, Ohio, 1928.

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Cupola Charge

The cupola charge varies greatly in different plants. As Fuel is one of the major constituents of the charge and probably the most expensive, pound for pound, its consumption is of great economic importance. In this connection it is interesting to note the great variation in fuel ratios as used in the plants visited. In order to avoid exposure of individual operations, ratios only are given, without key to plants.

Variations in Fuel Consumption at Different Plants

(Tons slag or rock melted per ton of fuel used)

Rock plants	Slag plants
3	3.5
1.3	2
5	1.9 1/
2.1	3.7
2.8 2/	2.8 2/

1 - Includes a small addition of coke.
2 - Unweighted average of four plants.

These rather wide variations in fuel efficiency obtained by different operators indicate the need for careful study of this problem.

One operator reports maximum and minimum limits for wool rock analyses to produce good wool as follows:

	SiO_2	Fe_2O_3 Al_2O_3	CaO	MgO	CO_2
Maximum . . .	34.22	11.54	15.80	10.98	21.90
Minimum . . .	28.28	10.00	23.10	8.68	18.14

Typical analyses of slags used in making mineral wool are as follows:

SiO_2	Fe_2O_3	Al_2O_3	CaO	MgO	S
38.0	1.0	11.0	28.0	19.0	.5-.8
38.4	0.7	10.5	31.5	15.3	1.6

An analysis of rock wool as obtained some time ago was as follows:

SiO	Al_2O_3 Fe_2O_3	CaO	MgO	S	Undetermined
42.84	1.40	51.74	2.75	0.0	1.27

While slag, either by itself or mixed with limestone, and natural wool rock are the two principal raw materials used, at least one manufacturer produces wool from silica rock and calcite. By proper chemical control, wool can be made from a number of materials.

Temperature Control

The fusion temperature obtained in the cupolas is reported as ranging from 2,800 to 3,000°F. There are no data as to whether the temperature or the time required for fusion varies with different raw materials. One operator, however, reports that wool rock obtained on the surface of the quarry is easier to fuse than that at some distance below the surface. At this quarry the surface rock presented a yellowish color while that below was blue, indicating that changes due to alteration by descending surface waters may have a bearing on fusibility.

Blowing Wool

Irrespective of the temperature at which the materials fuse, the temperature or at least the fluidity of the issuing stream of slag is of the utmost importance. The quality of the resulting wool with regard to amount of shot contained and length and flexibility of threads, depends on the temperature of the slag stream when it encounters the high-pressure steam. Careful research into several phases of this step in the operation will undoubtedly prove fruitful. Mention may be made of the following:

1. The relation between the slag temperature and the length of fiber.

2. The relation between the size of the slag stream and the quality of the wool.

3. The relation between the temperature of the slag and the temperature and pressure of the steam.

4. The relation between the quality of the wool and the air currents set up in the wool room by the blowing operation.

One plant visited several years ago by the author used compressed air instead of steam in blowing a slag wool. The wool produced exhibited extremely long fibers, but it is not known whether this was due to the use of air instead of steam or to the composition or temperature of the slag stream.

Collection of Blown Wool

Recent progress in the industry indicates a decided trend toward mechanical and automatic handling of the product and a decrease in manual labor. This result is in line with the mechanical progress of the times; however, while considerable progress has been made, there are many problems yet to be solved. It is interesting to note the various ways in which different operators are using automatic conveyors to eliminate hand collection.

Production and Productive Capacity

Available production statistics are summarized in Mineral Industry for 1911 as follows:

Production of Mineral Wool in the United States

(In tons of 2,000 pounds)

Year	Amount	Value	Per ton	Year	Amount	Value	Per ton
1900	6,002	$60,320	$10.05	1906	5,375	$55,550	$10.33
1901	6,272	68,992	11.00	1907	9,008	81,769	9.08
1902	10,843	105,814	9.67	1908	9,197	77,228	8.40
1903	(1)	- - -	- - -	1909	11,626	101,621	8.74
1904	(1)	- - -	- - -	1910	8,408	84,012	9.99
1905	6,164	69,560	11.28	1911	7,514	65,500	8.72

(1) - No statistics collected.

Although no actural figures for mineral-wool production appear to have been collected, the industry seems to have grown considerably since 1911. The 24 cupolas listed in the eight plants now operating have a potential capacity of 24,000 pounds of wool per hour. On the basis of a 200-day year, and allowing 24 hours per day for 22 cupolas and 16 hours daily for the two brick cupolas, the total annual production capacity can be calculated at roughly 54,400 tons. While this estimate is doubtless excessive, the output is evidently several times greater now than in 1911.

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Circular 6143
July, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SAFEGUARDING ELECTRICAL EQUIPMENT USED IN GASSY MINES [1]
EUROPEAN PRACTICE: III - GERMANY

By L. C. Ilsley[2]

Cooperation between the United States Bureau of Mines and the Safety in Mines Research Board of Great Britain, continuous since 1924, has made possible this and other papers on safety subjects. Grateful acknowledgment is made to representatives of the Board for their assistance in arranging visits to several mine safety stations, and to F. R. Wynne, Deputy Chief Inspector of Mines, for arranging visits to mines in Great Britain and other countries in Europe.

During the summer of 1927 the writer had the privilege of visiting the mine safety testing stations in great Britain, Belgium, Germany, and France, in the order named. All of these countries have large coal mines, many of which are rated as gassy. Therefore, when the installation of electrical equipment is contemplated, each of these countries is confronted with the same safety problem as is the United States - the development of electrical equipment that will not ignite gassy atmospheres which through neglect or accident might surround the equipment. The means employed by these countries in safeguarding gassy mines should therefore be of general interest to safety engineers in American coal mines, and it is proposed to give a brief survey for each of the four countries mentioned. The third of these surveys covers conditions in Germany.

GERMAN PIONEER RESEARCHES

Any attempt to survey the subject of explosion-proof equipment from an international standpoint should include a review of the pioneer work done by Germany in investigating the possibility of using electrical equipment in gassy mines and in outlining conditions under which such equipment might safely be introduced. The permissibility work connected with the investigation of electrical motors carried on by the United States Bureau of Mines has been influenced by the pioneer work of Dr. Beyling. His report,[3] an illustrated treatise of 89 printed pages, was translated and has been used as a reference work.

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1 The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be used: "This paper represents work done under a cooperative agreement between the U.S. Bureau of Mines and the Safety in Mines Research Board of Great Britain. Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

2 Electrical engineer, U. S. Bureau of Mines.

3 Beyling, C., Versuche zwecks Erprobung der Schlagwettersicherheit besonders geschetzter elecktrischer Motoren und Apparate sowie zur Ermittlung geeigneter Schutzvorrichtungen fur solche Betriebsmittel. Ausgefuhrt auf der berggewerkschaftlichen Versuchstrecke in Gelsenkirchen, 1906, 89 pp. Zeitschrift Glückauf, Essen, Jahrgang 1906, Parts No. 1-13.

Review of Beyling's Report

A report of the early research work conducted by Beyling was published during 1906 and is divided by him into three sections, as follows:

1. Experiments of the year 1903, in which tests were made using apparatus submitted by the electrical companies.

2. Experiments of the year 1904, in which tests were more of a research nature, involving the finding of a suitable means of protecting electrical motors and apparatus and testing different experimental devices.

3. Experiments of the year 1905, in which tests were made of motors having incorporated within them protective devices that had given promise in the experiments made during the preceding year.

The tests made during the first year showed the inadequacy of existing designs and the necessity for much experimental work.

The tests made during the second year are especially interesting to manufacturers who are designing or testing permissible-type equipment, in that nearly every conceivable scheme of protection was tried out during that year's experimental work. The tests covered the completely closed casing, wire-gauze-protected casing, the labyrinth casing, the casing with pipe protection, the flanged casing, the plate-protected casing, and the oil-filled casing.

The conclusions with respect to the various types of protection are of interest in that so many of them still hold good after 25 years' experience with the explosion-proof type of equipment. A free rendering of the more pertinent conclusions follows:

Specific Notes

> Totally enclosed compartment.- The totally enclosed casing for electric motors and apparatus offers a few special advantages. The fire damp can not so easily reach the protected and dangerous parts, as a longer interval of time is required for this purpose; the danger of an outside ignition is decreased thereby, particularly in the event of a sudden occurrence of large quantities of fire damp. The casing is also very resistant to external damage - that is, to fall of rocks or rough treatment. Finally, it protects the enclosed parts against moisture and dust. Against these advantages there is the disadvantage of lack of ventilation. Even if one or several holes were made therein, or if other openings were provided, the discharge openings, in order to be safe, have to be so small that no ventilation in the proper sense of the work takes place. If, therefore, one wishes to protect the coils of motors and of transformers and the resistances of starting and of regulating devices by means of closed casings, one must reckon with a considerable drop in the efficiency

of this apparatus, owing to poor ventilation. Therefore, these apparatus must have greater dimensions than would otherwise be necessary under the same circumstances; the initial cost and the cost of operation are thus increased. In order to obviate this undesirable feature, a different method of protection from fire damp is needed - that is, one that allows the heat to escape from the protected parts. If one rejects oil protection, there remains, for the attainment of this object, only a ventilated casing; that is to say, the protected space must communicate with the outside air through larger openings. But in order that the flame of a fire damp ignition occurring within this space should not communicate itself to the outside gases, care must be taken that the gases cool off sufficiently prior to their escape. The cooling can be accomplished in different ways. Such casings form a contrast to the totally enclosed casing in that the protective action of the totally enclosed casing is based on the fact that the hot combustion gases of a fire-damp explosion are retained in the casing as far as possible whereas the gases can escape from the casings with other protective devices after having been rendered harmless.

Gauze-protected compartments.- The experiments show that the wire-gauze-protected casing can be used to protect electric motors and apparatus against the danger of fire damp, if no intense ventilation prevails in the enclosed space. If such ventilation is present one must reckon with an intense after-burning which may lead to outside ignitions. The ventilation caused by rotation is especially harmful when the gauzes are near to or form part of the revolving portions.

The widespread view that wire gauze, which is an excellent protecting device for mine lamps, can also be used as a safeguard for electric motors and apparatus has not been corroborated to the full extent by the experiments made therewith, but, on the contrary, the wire-gauze casing presents some difficulties and does not always afford safety. Moreover, the gauzes may be easily damaged by external causes and become dangerous. In view of these defects an endeavor has been made to find another more adequate casing which possesses the advantage of the wire-gauze-protected casing - namely the ventilation of the enclosed space - without being beset by its drawbacks. Since ventilation was desired, only such casings were considered as provided communication from the interior to the exterior of the housings through large openings; the task was to provide some method of cooling the escaping gases other than that provided by the cooling action of wire gauze.

Labyrinth protection.[a]- The labyrinth casing does not seem to be suitable for the protection of electric motors and apparatus. Besides, it does not allow any appreciable ventilation of the enclosed space, owing to the smallness of the passage and the resistance met with by the cooling air on passing through the plates.

Flange-protected casings (slightly separated joints).[a]- The flange-protected casing (having an air-gap between two wide flanges) has no practical value for the protection of electric motors and apparatus, because it does not allow any appreciable ventilation of the enclosed space. The observations however, made on the occasion of the tests performed with this protective device, were of great usefulness for the further tests which aimed at finding a suitable substitute for the wire-gauze-protected casing.

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a The conclusions under headings "Labyrinth Protection" and "Flange-protected casings (slightly separated joints)" should be considered only from the objective of the experiments, which was to find a compartment which would give considerable ventilation and pressure relief. Both of these types of protection are used quite extensively in Germany.

Plate-protected casings.- The plate-protected casing should be a substitute for the wire-gauze-protected casing. It shares its advantages without being beset by its drawbacks; for this reason a short comparison between the two seems advantageous.

There is a certain agreement between the two casings in their action. In both the ignited gases can escape in a finely subdivided condition through narrow openings and must flow along metallic walls which give them an opportunity to cool off. The advantage of the plate-protected casing consists in the fact that the mass of the heat-absorbing body is much larger, and that the discharged gases must remain in contact therewith for a longer time than in the case of the wire-gauze-protected casing. In consequence thereof it still affords safety if the pressure and hance the velocity of the hot gases is high and the absorption of the cooling body is very large. But since beginning at a certain pressure the cooling action is supplemented or replaced by other factors, the safety of the plate protection is therefore unlimited, whereas the wire-gauze enclosure fails at a certain pressure which varies with the number of layers.

The other drawbacks of gauze-protected casing either do not exist or are present to a slight extent in the plate-protected casing. Afterburning occurs seldom and is of short duration. Its consequences are therefore less to be feared; it can not cause incandescence and excessive heating of the protecting metallic plates. The plate protection is much better safeguarded against external damage by its greater mechanical strength; in good designs, even small punctures, which may occur through injury to the wire-gauze, are impossible. For this reason, the plate-protected casing requires less supervision.

The most essential advantage of the wire-gauze-protected casing, the attainment of good ventilation, can also be obtained by the plate-protected casing. The plate protection must of course have larger dimensions if the ventilation is not promoted by artificial means.

As in the wire-gauze protection, excessive openings proved to be dangerous in the plate protection also, inasmuch as the protection depends on the cooling effect. An increase of 1 millimeter in a single slit was enough to bring about outside ignitions. Therefore, if such devices are to be practicable, great care must be taken in their manufacture. If the device is equipped with but a small number of plates, notice must be taken that a high pressure is capable of arising within the casing. Hence the casing walls must receive a corresponding thickness. The strengthening of the casing walls is also required on account of the high pressure which may arise in consequence of subdivisions of the enclosed space or in consequence of rotation. In order to prevent, as far as can be done, the occurrence of a high pressure in the casing, the discharge opening should be made as large as possible. In addition to the arrangement of numerous plates, this is attained by enlarging the inside diameter of the plates to the greatest extent allowable.

The plate-protected casing affords absolute safety if the plates are not more than 0.5 millimeter (0.02 inches) apart, if the width of the flanges amounts to 50 millimeters (1.97 inches), the thickness to 0.5 millimeter (0.02 inches), and if all the parts of the casing have such dimensions as to be capable of resisting high pressures.

The safety of plate protection is independent of the content of the enclosed space; of the magnitude of the total discharge opening, which in its turn is conditioned by the number of slits and the magnitude of the inside diameter of each plate; of the position of the ignition point; and of the composition of the fire damp mixture.

The larger the total discharge opening in comparison with the enclosed space the greater is the effective cooling action of the plate protection and the smaller is the pressure arising in the casing.

Openings in the casing, if larger than the slits, are dangerous.

The afterburning of the fire damp seldom occurs, and has a long duration only when the enclosed space is ventilated artifically with great intensity. The safety of the casing is not harmed thereby.

By means of the plate protection one may obtain a practical, useful and absolutely explosion-proof casing for electric motors and apparatus. With further improvement and test of this protective device, one could consider the problem, which was proposed at the beginning of the fundamental tests, as solved.

General Conslusions

Beyling's report closes with a set of conclusions based not only upon the experiments made using various types of experimental protective devices, but also upon later tests made with motors and other apparatus equipped with protective devices built as a result of the earlier experiments. These final conclusions, though rather extended, are so much to the point that it has seemed best to quote them in their entirety:

> Various Types of Casing Discussed.- Of the various protective devices for electric motors and apparatus which had been subjected to a detailed test, the labyrinth, the pipe, and the flange-protected casing, proved to be insufficient or unsuitable. The following have been found useful on the basis of the tests: The closed casing, the wire-gauze-protected casing, the plate-protected casing, and the oil-filled casing.
>
> The totally enclosed casing can be used only for apparatus that does not become considerably heated and does not require, therefore, any ventilation of the enclosed space. As it must be made very resistant to explosion pressures, it is heavy, but it offers the advantage of being very resistant to external injury.
>
> The wire-gauze-protected casing allows very good ventilation of the enclosed space; it is cheap and light. The mechanical resistance of the wire-gauze area is very small, and the consequences of afterburning are harmful in the case of artificial ventilation.

The plate-protected casing likewise gives the possibility of ventilating the protected parts; but such ventilation, in the absence of an artificial one, requires large dimensions of the protected device and hence entails a higher cost than in the case of the wire-gauze protection. Its mechanical resistance, however, is incomparably greater, and its safety is not endangered by afterburning.

The oil-filled casing does not allow the mine gases any access to the sparking parts, and therefore affords the best protection. Its field of applicability is limited, however.

The most important deductions from the tests with regard to the practical construction of these casings, and those which must therefore be taken into consideration in the construction of explosion-proof motors or apparatus, are as follows:

Details covering construction.- For all casings except the oil-filled casing too great subdivision of the enclosed space, and particularly the use of several larger spaces connected by constricted passages, are to be avoided. Care should be taken lest such connections be made through the oil chambers of motor bearings.

The joints between the various parts of the casing, as well as the surfaces of contact of lids, doors, covers, etc. should be built in the form of wide, smooth flanges.

It is not advisable to use rubber, asbestos, or other such material for gaskets. If, however, gaskets are used, they must be placed in such a way that they will not be driven out by the high pressure arising within the casing.

Shafts and spindles should traverse the casing wall through long metallic bearings (50 millimeters, if possible) (1.97 inches) which in turn should be solidly connected with the protecting casing.

The same holds for the passage of connection cables; the lead entrances should be sealed with insulating compound. Rubber bushings should be avoided.

Hollow shafts should be filled with insulating compound.

All screws should be so secured that they do not become loosened in operation and thus give rise to through-holes.

In the completely closed casing all parts should be able to resist a pressure of 8 atmospheres. Holes for the release of pressure should be avoided.

In the wire-gauze-protected casing standard, flame-safety lamp wire-gauze with 144 meshes per square centimeter and wire 0.35 millimeter (0.014 inch) in diameter should be used. Wire-gauze with a larger width of the meshes or smaller thickness of the wire should be avoided.

The wire gauze should be made of bronze or zinc-coated steel.

It must be uniform, free from flaws, and clean.

The total protecting gauze area (the sum of all superposed gauze areas) should be at least 150 square centimeters per liter (658 square inches per cubic foot) of gas content of the enclosed space.

The wire gauze should consist of at least two layers.

The distance between the gauzes should not be less than 5 millimeters (0.20 inches) and not more than 20 millimeters (0.79 inches).

Larger gauze areas should be provided with reinforcement.

The wire gauze should be arranged as a removable part to allow easy inspection and replacement of gauze.

All openings in the casing should be carefully avoided.

The gauze areas should be so placed that the afterburning flames do not strike the gauze, and so that combustible substances can not fall upon them.

The wire gauze should not be fastened by means of soldering. It should preferably be stretched in substantial frames or similar devices and bolted securely.

The wire gauze should be protected against external damage by perforated plates or similar means.

Plate-protected casings.- In the plate-protected casing the annular metallic plates should have flanges 50 millimeters (1.97 inches) wide and by 0.5 millimeter (0.02 inch) thick and should be so arranged through the insertion of suitable spacers that the maximum separation (width of slit) between them does not exceed 0.5 millimeter (0.02 inch).

The assembling of the plates should be done very carefully. No slit should be wider than 0.5 millimeter (0.02 inch).

The material used should be bronze, brass, tin-coated steel, or zinc-coated steel.

All open holes should be avoided. The interior of the enclosed space should be connected with the outside air only through the slits between the plates.

To avoid excessive internal pressure, the number of the plates (slits) and the inside diameter of the plates should be made as large as possible.

If the total discharge opening is made small in comparison with the gas content of the casing, then all the parts of the casing should be strong enough to be capable of withstanding a pressure of several atmospheres.

The plate assembly should be placed in removable parts so as to make convenient inspection and replacement of plates possible.

The plate assembly should be protected against external injury by a cover.

Oil-filled casings.- The oil casing must be liberally enough dimensioned that escape of sparks above the oil level is prevented.

The required height of the oil level should be determined by the manufacturer by means of a practical test and indicated by him by a special mark.

The height of the oil level should be evident without opening the casing.

Contacts or other current-carrying parts which are immersed in oil must be such as not to cause an intensive decomposition or gasification of the oil when the circuit is closed or opened.

The casing should be made in such a way that the sparking parts are not laid bare in consequence of excessive movement of the oil.

If the escaping of the sparks from the oil can not be prevented, then the oil box itself should be enclosed according to the principles of a closed casing.

These points should serve to indicate how the casings are to be made in order that the particular protective devices on which their safety is based should be efficient.

General consideration.- In addition to this, there are a number of construction devices which contribute to increasing safety in other ways. Such, for instance, are the interlocks designed to allow the opening of the protected casings only when the spark-producing parts are cut out of the circuit. A similar precaution requires the use of lead seals, so that only specially authorized, qualified persons can open the casings. To prevent short-circuit sparks, care should be taken that no bare wires or terminals should be placed outside of the casing. The casing should be made strong enough so as not to be broken by falling rocks.

The casing should not be made too small. For the smallness of the enclosed space hardly contributes toward increasing the protection against fire damp, it renders difficult the exchanging of brushes and the access to any other enclosed parts. The casings should also be provided with solid and tight glass windows which will allow the interior to be observed without opening the casing.[b]

In actual operation, the protected casings should be maintained in the condition which conduces to their safety. Particularly the wire-gauze-protected casing requires close supervision. The gauzes should be kept clean and should be inspected frequently. In the oil-filled casing the height of the oil level should be observed regularly; care should also be taken to have suitable oil. The plate protection should be protected against dangerous changes by its inherent strength; nevertheless, dust and moisture should be removed as much as possible. The closed casing requires the least attention.

Selection of casing for specific service.- Finally, let us discuss briefly the question as to what kind of casing is to be selected for each electric apparatus.

Small motors such as those used for drilling machines and small coal-cutting machines should be equipped with a totally enclosed casing. They are also protected thereby against the adverse external conditions to which they are exposed.

In the case of larger motors, such as those used for driving auxiliary ventilating fans, larger coal-cutting machines, portable air compressors, auxiliary pumps, and hoists, it is advisable as a rule to enclose the windings and, in particular, the sparking parts.

For the latter are more likely to cause ignition of fire damp than the coils; they can also be more easily protected against fire damp because they do not need any ventilation. The closed casings are suitable for the protection of motors up to a certain size. Beyond this the wire-gauze or the plate-protected casings are to be preferred because they are lighter. For the frame of the motor, it is best to use the plate protections.

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b Observation windows are no longer used. The windows referred to here mean the gauge glasses for oil-filled casings.

In the mine workings proper, motors up to 30 horsepower together with the apparatus belonging thereto, are commonly used. As these are the most exposed to the danger of fire damp, especial attention has been paid in our tests to the consideration of equipping such apparatus with explosion-proof casings. But the protective devices which have been discussed can also be used for motors up to 50 horsepower.

Still larger motors are used principally for pumps, hoists, larger fans, and compressors and are installed near the downcast shafts or near underground places that receive fresh air. It is therefore hardly necessary to equip these motors with an explosion-proof casing. If, however, such a protection is desired, it will undoubtedly be sufficient to protect the sparking parts; this can be done by means of the wire-gauze-and-plate protection. The enclosing of the windings of very large motors would meet with considerable difficulties. It is not necessary, however, because the ignition of the fire damp through the windings is almost out of the question. In the first place the motor is situated in the fresh air current; it could not, therefore, be surrounded by mine gases except in the case of an extraordinary disturbance; even then the windings would have to be injured at the exact instant this unusual accident occurred. Such a coincidence of events need not be provided for.

For apparatus that does not require any ventilation, such as switches and fuses, it is best to use the closed or the oil-filled casing. The contacts of starting and regulating resistances should be protected by a closed casing. Resistances which must dissipate heat may be immersed in oil in the case of starting resistances, but regulating resistances should preferably be ventilated by wire-gauze or plate protection.

As a rule, the oil-filled casing will be sufficient for transformers.

So far as the danger of igniting fire damp by electric motors and apparatus is concerned, there are no longer any difficulties in the way of completely utilizing the great advantages offered by the electric power transmission for underground work, particularly in the mine workings proper. Thus, despite the many initial failures, the object aimed at has finally been attained by means of tedious and often dangerous tests.

GERMAN REGULATIONS FOR THE CONSTRUCTION OF FIREDAMP-PROOF ELECTRIC MACHINES AND APPARATUS

The regulations of 1926 cover the essential requirements which electrical equipment had to meet at the time the visit was made. These requirements are included among other electrical regulations in a booklet[4] prepared by Dr. C. L. Weber and published by Julius Springer of Berlin. A free rendering of these regulations follows:

Regulations for the Construction of Firedamp-Proof
Protective Devices in Electrical Machines,
Transformers, and Apparatus. (Effective January 1, 1926.)

4 Weber, C. L., Erläuterungen zu den Vorschriften für Errichtung und Betrieb von Starkstromanlagen einschl. Bergwerksvorschriften. Published by Julius Springer of Berlin, 1927.

Paragraph 1. All machines, transformers, and apparatus which are intended for use in gaseous places in mines must meet the existing regulations, rules, and standards[c] of the Association of German Electrical Engineers, as far as is not otherwise stated in the exceptions which follows.[d]

Paragraph 2. All parts of electrical machines and apparatus, which give rise to sparking in normal operation, must be enclosed in firedamp-proof casings. The following casings are considered firedamp-proof

(In addition to the classes described in (A), (B), and (C), wire-gauze enclosures used as in the Davy safety lamp were permissible up to this time. These enclosures do not protect machines, etc., because the wire gauze may get dirty or be destroyed by blows or by rusting. Gauzes have therefore been abandoned.)

(A) Sealed compartments.- Totally enclosed compartments must meet the following requirements:

1. In the case of machines or apparatus having an air capacity greater than 1 liter, all parts of the enclosure must be designed for a pressure of 8 atmospheres above atmospheric pressure; those having a small air capacity, for a pressure of 3 atmospheres above atmospheric pressure;[e] subdivisions of the enclosed space which are connected by small openings and therefore may be subjected to excessive pressures should be avoided.[f]

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c In addition to these construction rules the following should be noted: Rules for the design and testing of electric machines, R.E.M./1923; Rules for Rating and test of transformers, R.E.T./1923; Explanatory notes on the construction and testing of alternating current high-tension apparatus for 1,500 volts nominal voltage and upward; Rules for the construction, testing and use of switching apparatus for up to 500 volts alternating current and 3,000 volt direct current, R.E.S./1928; Regulations for insulated circuits in power installations, 1926; Regulations for the construction and testing of installation material, 1926.

d For the scope of electrical operation in mining see Phillippi, ETZ, 1925, p. 997; and Glückauf, 1925, p. 807. The permissibility of specific types of applications, machines, and apparatus is determined by the mines officials (see par. 12).

e The above specified resistance to pressure is not required in the sense that the specified pressure must be withstood continuously during operations, so that an air-tight seal would be required; but the enclosure must withstand a sudden pressure caused by an explosion without being broken or having its fastenings torn.

f The air gap between the stator and rotor does not act as a small opening because of its considerable total area.

2. The joints of those parts of compartments and housings fitting against each other, as well as the joints of lids, doors, and covers, must be formed by broad smoothly machined flanges.[g] Gaskets are preferably avoided; in case they are used, they must be applied in such a manner that they can not be pushed out by the force of an explosion. Gaskets of rubber, asbestos, and similar perishable material are not permitted.[h] Screws, rivets, etc., used to fasten on such covers must not pass through the walls of the housing but must end in bottomed holes.[i] The fastenings of the lids must be so secured that they can not loosen in operation and can only be opened with special appliances.[j]

3. Shafts and spindles shall be carried through the casing walls in long metal busings firmly attached to the housing.[k] Lead entrances must be so sealed that they will safely withstand the pressure of the explosion.

(B) Plate-protected casings.- Plate protection functions because the gases forced out after an internal ignition by the resulting expansion are cooled to so low a temperature in passing through the plates that the ignition can not be transmitted to the outside. On ignition, the internal pressure can not become so great as in totally enclosed compartments. Still, one must count on a certain pressure, and therefore sufficient strength of housing parts and their safe assembly must be provided. This is especially emphasized by the ruling under section B3. Plate protection is especially suitable for machines and apparatus of large internal volume - for storage battery boxes, for example. Plate protection consists of arranging stacks of metal plates in openings in the housing; the plates are maintained at definite intervals by spacers.

Plate-protected casings must conform to the following requirements:

1. The metal plates must be at least 50 millimeters (1.97 inches) wide and 0.5 millimeter (0.02 inch) thick; they must be so fixed by suitable spacers that the intervals do not exceed 0.5 millimeter (0.02 inch) and can not be increased by bending of the plates.[l] Plates subject to rust are not permissible.[m]

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g The width of flanges for closely fitting surfaces can be less than for those not closely fitted. In the first case 8 millimeters is regarded as the minimum width, in the second case 8 millimeters for spaces up to the 1 liter, and 25 millimeters for greater spaces.

h Care must be taken to prevent the pushed-out packing from opening up a broad crack through which an explosion occurring inside of the enclosure can be transmitted to the outside. As is explained in note e of paragraph 1, an air-tight enclosure is not required. Narrow cracks (slits) can act as a relief for the pressure of explosions, but like the plate protection must prevent the transmission of ignitions to the outside of the compartment.

i The omission of a screw shall not cause a through-hole in the wall of the enclosure.

j The "special appliance" shall not be accessible or available to every miner, but only to authorized persons. The device must therefore be, for example, a special key. (Lead seals are not sufficient.)

k Hollow shafts passing through the housing walls must have their bores suitably sealed.

l The metal of the plates must have sufficient rigidity, and individual slits should not be too long. The total area of all the slits must be as large as possible.

m Zinc-free aluminum bronze is, for instance, nonrusting material.

2. The plate assembly must be protected from external injury, and must be so assembled that it can be removed only with special appliances.

3. The conditions of (A), sections 2 and 3, must be fulfilled. Long continued high pressure can not take place with plate protection. The conditions of (A) 2 and (A) 3 must only be considered with respect to sudden excessive pressures.

(C) Oil protection.- Oil protection consists of building the whole apparatus - with respect to those parts liable to sparking or overheating due to current - in containers filled with resin-free and acid-free mineral oil.

These containers shall be designed with an oil level so high that the occurrence of sparks above the oil level is prevented. The necessary height of the oil level must be fixed by a mark. The height of the oil level must be evident from the outside.

Paragraph 3: For portable machines, transformers, and apparatus, oil-protection is not permissible. (Those installations which are fed by a flexible conductor are considered portable; not those which at certain intervals are moved to a new place and then again are permanently connected.)

The oil protection can be built very satisfactorily; nevertheless in portable appliances rough handling as well as accidental breakage from falls of roof, etc., must be considered. Ignition resulting from breakage of the oil reservoirs is especially to be feared because of filling haulageways with smoke.

Paragraph 4: Such parts of machines, transformers, and apparatus in which sparks occur or overheating may take place only in exceptional cases are built for greater safety than is required by standard specifications and especially are safeguarded as follows:[n]

1. By a special mechanical protection of the live parts against personal contact as well as against injury and short-circuit by foreign bodies.

2. By lowering the allowable maximum temperature rise in the previously mentioned regulations, rules, and standards by 10°C.

Alternating-current induction motors require an air gap 40 to 60 per cent greater than in normal design between stator and rotor. (See D.I.N. V.D.E. 2650 and 2651.)

Paragraph 5: In a.c. motors with short-circuited rotor, joints between bars and short-circuiting ring must be brazed or some other similar safe method used.

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n Starting resistances shall be so constructed by the proper choice of materials, proper dimensions, and design that they are protected from glowing temperatures or injury under all conditions.

Paragraph 6: Liquid rheostats are prohibited. (There is no sure means of protecting liquid rheostates under all conditions from evaporation with resultant sparking.)

Paragraph 7: In the case of metallic resistances, special safety devices may be omitted if, at the same time:[o]

1. The load on the material is so low that a dangerous rise in temperature is precluded.

2. The resistance material is rugged enough that it will not break in ordinary operation, and is firmly enough mounted that its parts can not short-circuit.

3. Foreign bodies or drops of water are prevented from falling in by a proper covering.

4. All wire connections are soldered or otherwise securely made.

Paragraph 8: All screw-fastened contacts - connection terminals of motors, resistances, etc. - which are not protected by enclosures must be so safeguarded that a loosening of the screw, resulting in poor contacts, can not occur.

Paragraph 9: Contact plugs must be constructed so that the plugs seat themselves tightly in the sockets and allow no sparks to occur while in the operating position. They must be interlocked with a firedamp-proof switch in such a manner that insertion and withdrawal of the plug is possible only when the connection is dead.

Paragraph 10: Fuse compartments must be combined and interlocked with a firedamp-proof switch in such a manner that cartridges can be taken out and replaced only when they are dead. Fuses can be used only in locked enclosures which must be made firedamp-proof to conform to Paragraph 2A, provided the plugs themselves are not firedamp-proof.[p]

Paragraph 11: Only rubber-covered cables of strong construction can be used as flexible conductors (NSH "Regulations for insulated circuits in power installations.")[q]

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o The rules for design and testing of starters, etc., limit the allowable rise in temperature. Here an especially reliable design is required.

p The opening for the tell-tales for fuses is sealed in the firedamp-proof screw plugs.

q All flexible cables used must be examined at regular intervals, and when defects are found must be immediately exchanged for entirely dependable cables. It is recommended for all gaseous mines that flexible cables be replaced after a specified period of use. In oil mines and oil shafts, rubber-covered cables without a special protective armor, are specially endangered since rubber is chemically acted upon by oil.

Paragraph 12: Machines, transformers, and apparatus other than of the described construction are permissible if they have been shown to be firedamp-proof by special test in an authorized firedamp testing gallery.[r]

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r Also machines, etc., meeting the preceding regulations are subject to the approval of the Department of Mines. The later determines in what manner the contents of the regulations are interpreted and which designs are permitted for particular cases. These regulations are not applicable without further expansion to operations in which other gases or vapors (benzine, hydrogen, etc.) than fire damp, give rise to specific hazards. They can, however, be used as the basis for such specifications.

GERMAN EXPERIMENT STATION VISITED

On September 2, 1927, accompanied by G. Allsop of the British Safety in Mines Research Station, J. A. B. Horsley, H. M. Electrical Inspector of Mines, and Herr Stoeck, Director of C.E.A.G. Work in Dortmund, the writer had the pleasure of visiting the German Mines Experiment Station at Derne, which is only a short distance from Dortmund.

Electrical equipment, flame safety lamps, and explosives are tested at this station. The flame safety lamp station is similar to those in Belgium and the United States. The lamp-testing gallery used in the Pittsburgh station was constructed from plans obtained in Belgium.

The gallery in which motors are tested is similar to the one built for the earlier tests at the original Ruhr testing station in Gelsenkirchen.

A motor or other piece of electrical apparatus is subjected to three tests in gas (methane).

Detailed drawings are required of any equipment that is to be submitted for tests, and a great deal of attention is given to the preliminary examination of these drawings. As shown by the regulations quoted, considerable information is given as to the construction for this type of apparatus, and equipment is carefully examined to see that it comes within the limits specified. The official gallery facilities in Great Britain, Belgium, and Germany do not appear to be so convenient as at the Pittsburgh testing station for making a large number of tests quickly.

REGULATIONS IN GERMANY AND THE UNITED STATES CONTRASTED

A study of the chief points set forth in the regulations reveals a very close agreement with the U. S. Bureau of Mines requirements for permissible equipment. There are a few minor differences such as the permitting of gaskets betwee joints, which the German regulation allows but does not encourage; the requiremen

of special tools for renewing the fastenings to all permissible compartments, which is a step farther than American requirements have gone; and the elimination of gauze-protected compartments, which are not encouraged by the bureau.

German practice resembles the American in one respect - that trolley locomotives are allowed underground. The conditions governing their operation are much more restricted than in this country, and they are little used.

In Germany, as well as the other countries visited, alternating-current equipment predominated in the mines, whereas in American mines direct current is used largely.

In the United States some States have not as yet required permissible-motored equipment and others are very lax in their requirements, whereas in gassy mines in Germany it is understood that the following procedure is necessary:-- (1) The mine operator makes official application for permission to have a motor for a certain use. (2) The motor manufacturer must furnish a statement that the motor he is supplying is the same as the pattern that has been tested. (3) Upon the arrival at the mine the motor is dismantled and inspected by a Government inspector. (4) After the motor is installed inside the mine it is again inspected by a second Government inspector who issues special instruction regarding its use and application.

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June, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

A GAS EXPLOSION IN A ROCK-DUSTED MINE [1]

By G. S. McCaa[2]

Introduction

Rock-dusting is an effective means of preventing coal-dust from propagating an explosion in a coal mine, but it will not prevent gas from igniting explosively and with much attendant damage locally. Ventilation must therefore be effective at all times, regardless of the employment of other explosion-prevention methods, if gas explosions are to be avoided. This paper describes a gas explosion which occurred in a bituminous mine that was well rock-dusted at the point of ignition and which exemplifies the value of rock-dusting, the necessity of maintaining effective ventilation, and the need of permissible rock-dusting machines receiving power from a permissible locomotive.

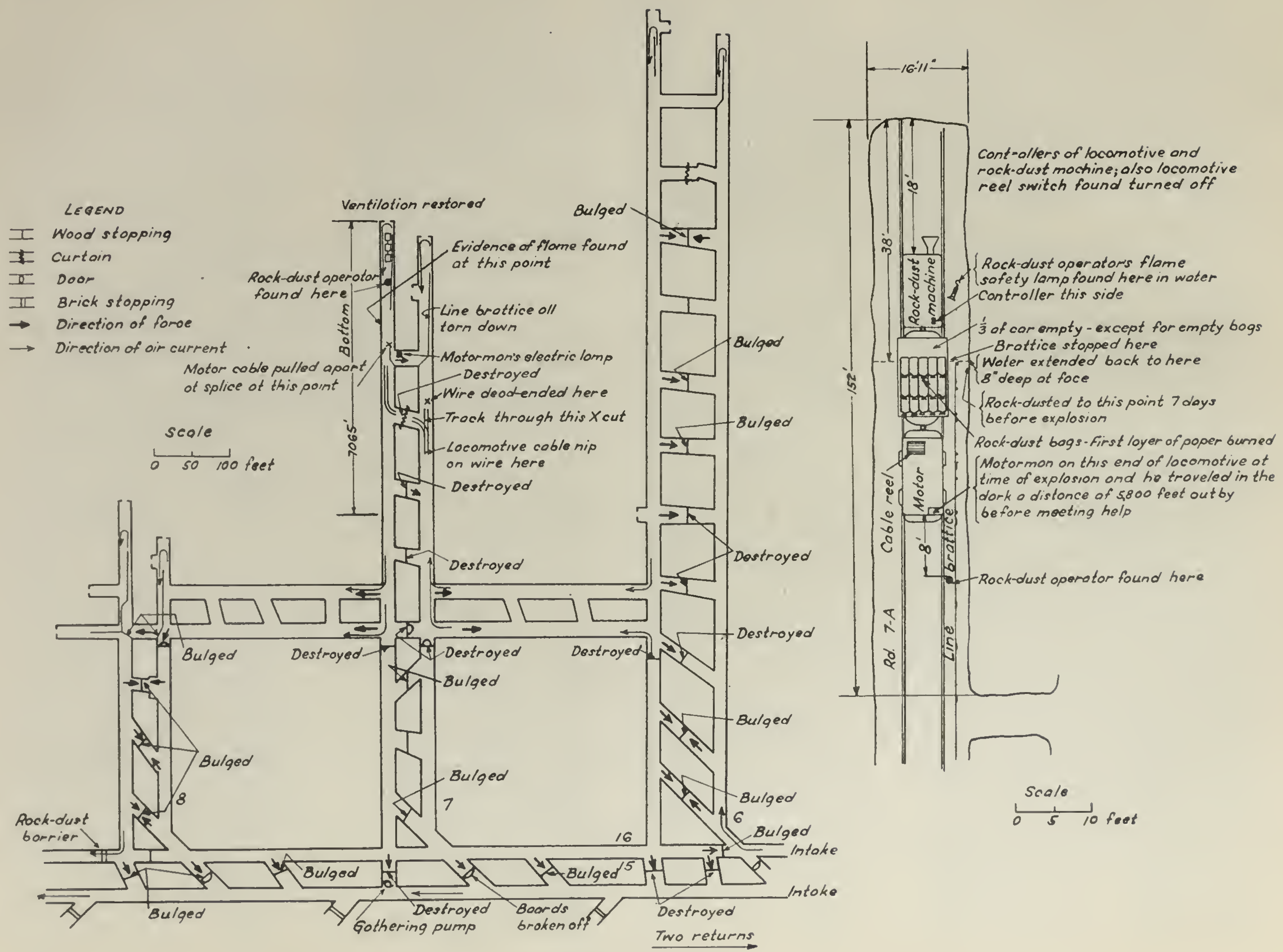
Description of Explosion

The mine in which this explosion occurred is developed on a panel system in a practically level seam; the panel entries are driven in pairs, and the air current intakes on the right entry and returns on the left entry. Near the mouth of the intake a door is placed to control the ventilation; board stoppings are placed in crosscuts, except the second from the face, in which a brattice cloth curtain is hung, the last crosscut is left open and line brattices conduct the air current from the last crosscut to the faces. In No. 7 entry the last crosscut was 152 feet from the face, the line brattice being within 38 feet of this face, and the second crosscut was 250 feet outby the face of the back entry or return. Seven days prior to the explosion the intake and return entries were rock-dusted to within 38 feet of the face, and as the entry had advanced during the week, the rock-dusting was to be extended.

A rock-dusting machine was being pushed by a cable-reel locomotive with a car of rock-dust between the locomotive and the dusting machine. The rock-dust train was pushed through the controlling door near the mouth of the intake entry, and the motorman closed the door after passing through. The operator of the rock-dusting machine preceded the motorman and went to the face of the return entry to

1 The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be used: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

2 District engineer, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.



Effect of a gas explosion in
in a rock-dusted mine

test for gas, leaving the curtain in the next to the last crosscut open, as was customary. The locomotive followed, the motorman attaching the head hook of the cable to the trolley wire in the intake entry before entering the return entry, and leaving the curtain open. The motorman, observing the rock-dusting machine operator holding his flame safety lamp about waist high and motioning him to advance, continued to the top of a hill about 50 feet from the face; he then closed the controller, set the brake, and was about to notify the other man to start the rock-dusting machine when a bright flash occurred near the cable reel on the locomotive. He immediately attempted to open the reel switch in the cab end of the locomotive when there was an explosion. The controller was found open immediately after the accident. The motorman was knocked out of the locomotive by the force of the blast, was severely burned about the face, chest and hands, and lost his electric cap lamp. He traveled in the dark a distance of 5,800 feet before he met help and was taken to the surface for treatment.

The rock-dusting machine operator's flame safety lamp was found about 20 feet from the face, and he was picked up by rescuers about 65 feet from the face, indicating that he traveled about 45 or 50 feet after the explosion. His heart was still beating when the rescuers arrived and carried him about 325 feet to fresh air. Artificial respiration was applied for 15 or 20 minutes but failed to revive the injured man, who died shortly after reaching fresh air. The rock-dusting machine operator's face, hands, mouth, and tongue were badly burned and the left leg of his trousers, from knee to thigh, was burned off. The two rescuers tested for gas at the point where the injured man was found, but no gas was detected.

The force of the explosion destroyed all of the board stoppings and doors for a distance of 700 feet, and those in line with the force were destroyed for a distance of about 1,000 feet. Stoppings as far as 1,500 feet were caused to bulge. The flame, in so far as could be ascertained, was limited to a zone about 90 feet long in the return entry; the limitation of the flame was no doubt due to the fact that this section of the mine was well rock-dusted from a point 10 to 15 feet outby the assigned origin of the explosion, thereby preventing coal-dust from propagating the explosion, or at least very definitely lessening the violence of any dust propagation which may have occurred.

Factors Contributing to Explosion

Several factors contributed to cause the explosion. Evidently the man in advance of the rock-dusting train made the error, not uncommon with experienced and unexperienced men alike, of failing to test at the roof in the high point about 50 feet from the face; he did test at the face and motioned the motorman to advance, assuming the entry clear of gas. Failure to close the curtain in the second crosscut after the rock-dusting equipment passed through to the return entry may have been a contributing factor also, and it would appear probable that this curtain was open or partly open before the rock-dusting machine came into this part of the mine. With ventilation stopped, gas issuing from the strata at the face would tend to seek the highest point, which in this case was bout 50 feet from the face. A considerable body of the methane might accumulate in 10 or 15

minutes if the ventilating current was stopped, although it is probable that the curtain had been open for a longer period. Finally, a source of flame was introduced in the open-type cable-reel locomotive which was used in connection with the rock-dusting unit. The locomotive presented many potential points for producing arcs or sparks; however, it is believed from an inspection of the locomotive cable that a splice in the cable became short-circuited near the reel at the top of the locomotive, and being relatively near the roof, ignited the body of gas.

Cables on mining equipment, especially on cable-reel locomotives, are decidedly hazardous. A manufacturer's agent states that some mines in his district make an average of 1.7 splices per week per cable, and one company buys about 500 patented splices per month. Many splices are made in the mine during the shift; such repairs are generally hurriedly made, with the result that the cables pull apart while in service.

There appears to be a false sense of security on the part of rock-dusting crews; the presence of rock-dust allays their fears of gas and coal-dust ignitions, and the men take liberties with ventilation and open-type electrical equipment. There should be no relaxation in the vigilance necessary to keep a coal mine safe; the rock-dust distributor and the locomotive transporting it have practically no protection against gas and coal-dust at or before the time the rock-dusting is started; even when the dust is being applied, there is no protection against gas. Only permissible electrical equipment, including locomotives, rock-dusting machines, etc., should be allowed near the face of workings and on return airways, even while rock-dusting.

Rock-dusting is gradually becoming standardized. The mines in which the application was often as far as 200 or 300 feet from the face, have recognized the dangers of such an unprotected length of entry and are now making the maximum non-rock-dusted distance 40 feet from the face. The value of such a standard is admirably illustrated in this explosion, and it is earnestly hoped that all mines will soon be rock-dusted from within at least 40 feet of the face of all workings to the mine opening.

Conclusions

There was unquestionably a relatively large accumulation of explosive gas at the high point where the locomotive stood on No. 7 air course. There is no direct evidence to show that the canvas curtain across the haulage chute (second open crosscut from the face) to No. 7 air course was left open before the rock-dust outfit entered, but it is reasonable to assume that this was the case, since the gas would not have collected if the air current had been sweeping the face and traveling past this high point in the air course. There is no means of determining where the gas tailed out, but it is believed that the rock-dusting machine and locomotive were both in the explosive mixture which was undoubtedly ignited by a flame, presumably an electric arc, the exact location of which has not been determined. The fact that this section of the mine was well rock-dusted to within 38 feet of the face on No. 7 air course limited the extent of the explosion essentially to the range of the explosive gas, and coal-dust played little

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or no part in the propagation of the explosion. It is entirely probable that the amount of gas which exploded would have propagated a coal-dust explosion throughout the mine if the mine had not been well rock-dusted.

The essential factors involved in this explosion are as follows:

1. An accumulation of methane existed near the face of No. 7 air course.

2. The locomotive was not of a permissible type and could ignite gas from more than one source by sparking or arcing.

3. Defects were found on the trailing cable that would cause a "short."

4. The rock-dusting equipment did not remain on the intake side outby the haulage crosscut to No. 7 air course until the air course had been tested for gas.

5. There was evident failure of the rock-dust operator to properly examine No. 7 air course.

6. A lighted flame safety lamp was carried by the rock-dust operator, but apparently was not used effectively.

Recommendations

1. Rigid examination of any place into which electric equipment is to enter should be made with an approved flame safety lamp by a competent person; the electrical equipment should remain on the intake side outby the last permanent open crosscut to the parallel entry until the test is completed.

2. Only men who in the opinion of the mine foreman are qualified to handle a flame safety lamp should be entrusted with the examination of working places prior to the entrance of electric equipment.

3. Test should be made at least every half hour while machines are operating in places that may generate explosive gas.

4. The operators of all electric equipment should inspect such equipment at the beginning and the end of each shift for defective cables, loose connections, loose or missing bolts, or for any defect that might cause an accident; effective repairs should be made promptly when defects are found.

5. The rock-dust accumulations on the motor of the rock-dusting machine should be blown off every day.

6. Insulators should be provided for all cables where they pass through steel housings.

7. Cover plates should be kept on commutators.

8. The bottom chain guard should be kept on the rock-dust machine.

9. No temporary or makeshift electric connections should be permitted.

10. Fire extinguishers of suitable type should be a part of the equipment on all electrically operated compressors and locomotives - such fire extinguishers, for instance, as carried on mining machines in many progressive mines at present.

11. No open-type electric switches should be used.

12. Accumulations of oil and grease around motors should not be permitted.

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Circular 6145 June, 1929

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

MINING METHODS AT MINAS DE MATAHAMBRE, MATAHAMBRE, PINAR DEL RIO, CUBA.

BY

George L. Richert

Circular 6145
June, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AT MINAS DE MATAHAMBRE, MATAHAMBRE, PINAR DEL RIO, CUBA [1]

By George L. Richert[2]

INTRODUCTION

The Minas de Matahambre is situated on the north coast of Cuba, in the Province of Pinar del Rio, approximately 150 miles west of Havana. This mine is the only producer of copper in Cuba, and has been worked for the past 15 years. In 1921 the property was affiliated with The American Metal Co., Ltd., and has been worked by them since that time under the direction of Mr. D. D. Homer, General Manager.

Extracts from P. D. Wilson's "Preliminary report on the geology at Minas de Matahambre" have been freely used in this report.

GEOLOGY

The main mountain range in the Province of Pinar del Rio, the Sierra Organos, parallels the coast in a northeasterly direction a few miles south of Matahambre. The district itself is in a comparatively low, outlying, parallel range, composed entirely of sedinmentary rocks - shale and metamorphosed sandstone - of which a great thickness is exposed. Mineralization is found at various points in this range of hills for a distance of about 14 miles. No igneous rocks have been recognized at or near Matahambre.

Although there is not a great deal of vegetation, except in some of the arroya bottoms, the surface is normally covered with thick deep grass. Two or three feet of loosely cemented talus and detritus usually cover the hill slopes, so that outcrops in place are hard to find, even where the grass has been burned or worn off. Rock exposed in gulch bottoms and the rather infrequent projecting knobs and areas of more resistant material are all that are available for surface study. The hillsides are usually gently sloping, but some of the gulches are steep-sided, indicating recent rapid erosion by the torrential rains of the wet summer season. The high humidity of the six months during the summer and the rapid run-off and erosion have resulted in complete but rather shallow oxidation and unimportant secondary enrichment of the sulphide ore bodies.

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1 The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be used: "Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"

2 One of the consulting engineers, U. S. Bureau of Mines.

In one of the steep gulches, where the original discovery at Minas de Matahambre was made, rare residual primary sulphides may be found in the actual outcrop, mixed with copper carbonates and oxides, a little sooty chalcocite, and porous, cellular iron oxides. This is the only place where even stains of copper have been seen on the surface. Very complete oxidation extends in places underground to a short distance above the fifth level, a maximum of 200 feet below the surface. Important primary ore bodies with irregular sooty chalcocite pockets and layers were mined on the fifth level, but no definite extensive secondarily enriched zone between the chalcopyrite of the primary ore and the almost copper-free leached limonite gossan above appears to have existed.

The mineralization at Matahambre is definitely related to fissuring, which almost always cuts across the bedding. Only locally has it been influenced by the bedding structure, and rarely does ore make off along favorable beds for any distance away from the zone of fracturing. The ore which follows bedding is of minor importance to that which occurs in the fissure zones. On none of the fissures has there been much movement, and most of that probably took place in the period of postmineral readjustment.

While no intrusive rocks have been found in the region, it is highly probable that the fissuring was induced by highly compressive stresses developed during the cooling of the igneous rock that surely exists at depth and is the source of the mineralizing solutions.

PHYSICAL CHARACTERISTICS OF ORE AND ENCLOSING ROCKS

The ore bodies in the Matahambre mine occur as large irregular pipes, usually with lenticular cross section, of primary chalcopyrite locally associated with quartz and pyrite in three well-defined mineralized fracture zones. The main mineralized fracture zone, in which the most important ore bodies are found, trends N. 30° E. and dips 42 to 45° northwest. Figure 1 is a cross section of No. 14 ore body, and Figure 2 is a progress map of the 1,300 foot level showing the shape of the ore bodies. The other two zones, about 1,200 feet apart and at each end of the main zone, strike northwest and dip steeply northeast. At Matahambre, ore has been found in sedimentary rocks of a total thickness of over 1,600 feet.

In all Matahambre ore bodies, especially those of larger size, the footwall is usually quartzite and the hanging wall is thin-bedded shale. The footwall will stand well and gives no trouble. The hanging wall, because it is naturally weak and because of the dip of the ore bodies, will not stand for any length of time if exposed for any distance and at a height of over 10 feet. This characteristic of the hanging wall of the ore bodies, together with the rather flat dip, definitely fixed the method and procedure of mining the stopes; namely, the cut-and-fill system, with cuts of small height.

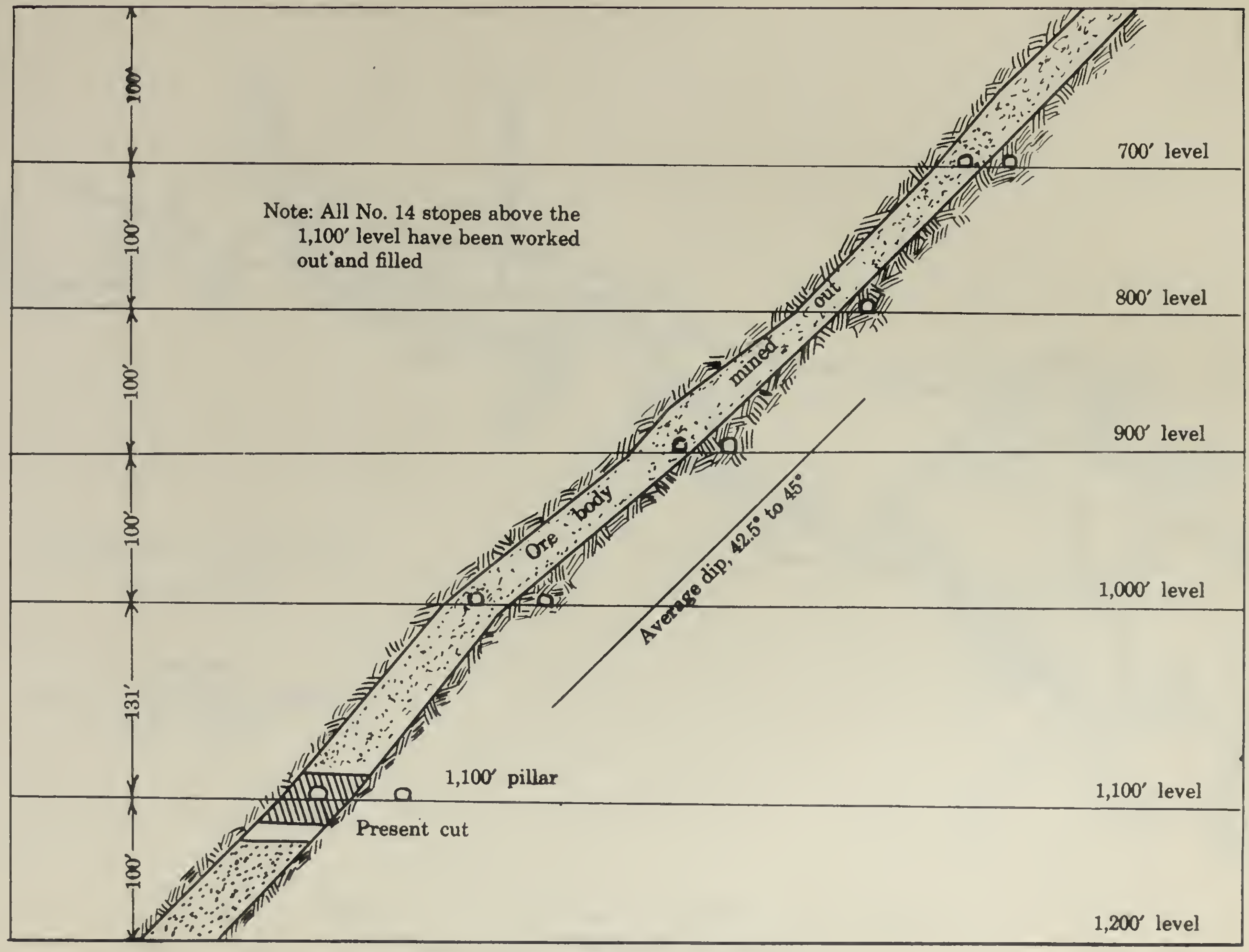

Figure 1.– East-west cross section of No. 14 ore body above 1,200-foot level

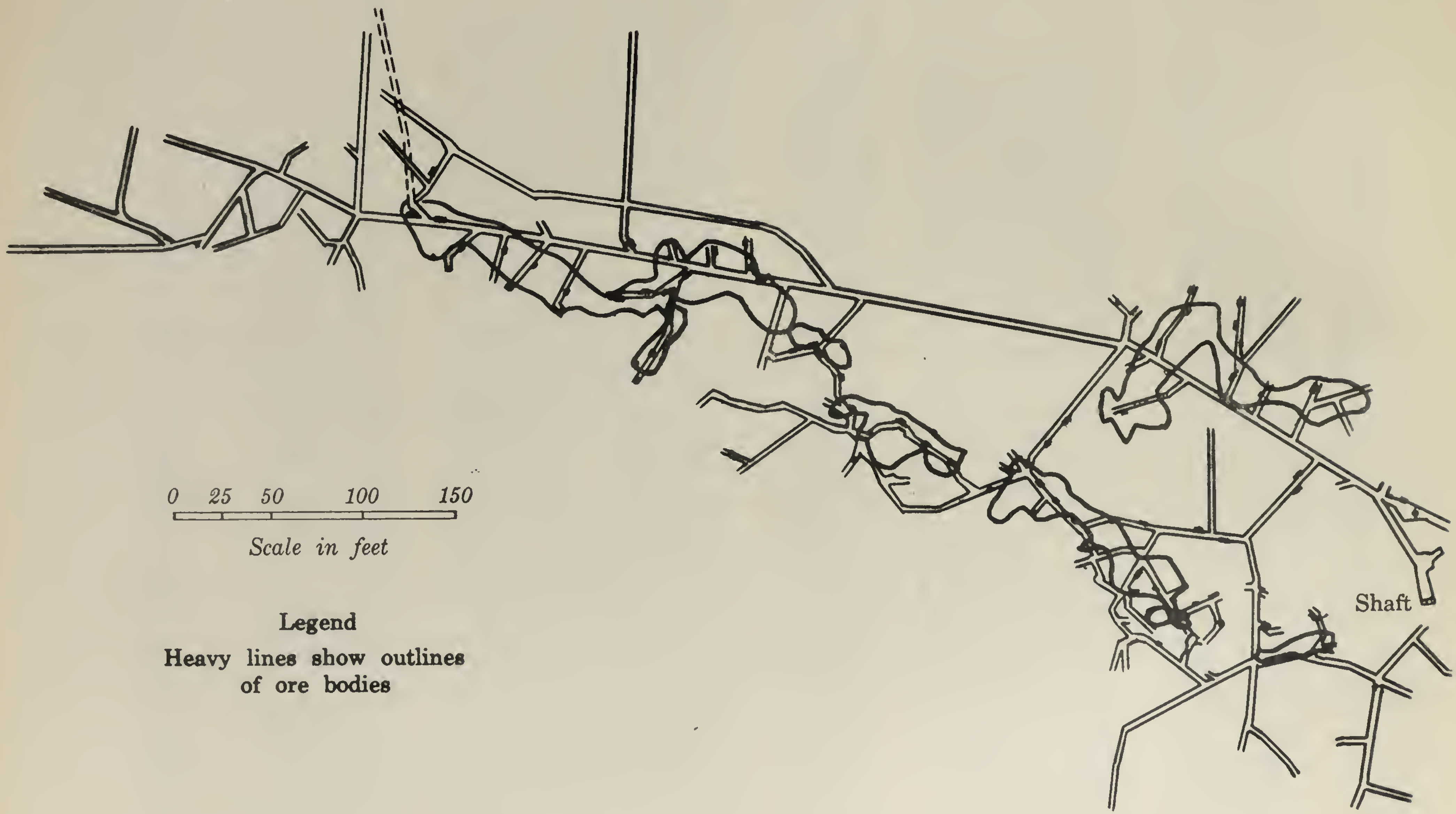

Figure 2.– Progress map of 1.300-foot level, showing outline of ore bodies and development

METHODS OF PROSPECTING AND EXPLORATION

As the dip of the main ore bodies is comparatively uniform, they can be projected with more or less accuracy from one level to the next. However, there are a number of small ore bodies, apparently offshoots of the larger ore bodies, which were found only by prospecting "likely country." At present, prospecting is carried on by diamond drilling, "deep-hole drilling," and by stope and level drifts and crosscuts.

Diamond drilling is carried on both from the surface and underground. On the surface, diamond-drill prospecting consists of drilling on the hanging-wall side of promising outcrops. As the outcrops have as a rule been thoroughly leached out so that only iron stains are left showing at the surface, the holes are drilled at such a distance from the outcrop that they should intersect the ore about 300 feet below the surface. As most of the large ore bodies did not outcrop on the surface, a number of holes were drilled where a prominent ridge or deep sharp ravine indicated faulting or possible movement.

Underground diamond-drill holes are put out where there is some indication of ore - stains or low-grade ore - not directly connected with a known ore body. These holes are as a rule flat and from 100 to 400 feet in length. Diamond drilling is also used when a drift or crosscut has opened some new and undeveloped country.

"Deep-hole Drill" drilling, "while comparatively new at Minas de Matahambre has proved valuable in prospecting for ore bodies which have flattened or steepened and have not been found in the locations projected from the level above. The "deep-hole drills" are also used advantageously in prospecting on the levels for possible extensions in the hanging wall or footwall of known orebodies, and are used in the stopes themselves when there is an indication that the stope has narrowed because of intrusions of waste or following false walls. When the ordinary drifting machine is used the average depth of these holes is 60 to 70 feet. With the large machine mounted on two columns, the holes average 150 to 200 feet. There are at present three "deep-hole drills" in use. Two crews use ordinary drifting machines and one crew uses a large special machine adapted to this work.

Stope drifts and crosscuts are used wherever there is a possibility that an apparently poor showing along the side of a stope may widen out or make ore. These drifts and crosscuts are also used whenever there is any question as to whether or not the true hanging or footwall has been reached. These prospects are usually from 4 to 15 feet long and are run after a cut has been taken from a stope and before the stope has been filled.

Drifts and crosscuts are used for prospecting when it is desired to open some new country for prospecting and the distance prohibits the use of drilling.

METHODS OF SAMPLING AND ESTIMATING TONNAGES

There is no regular system of sampling at Minas de Matahambre. After a short time the grade of the ore can be judged fairly accurately by the eye. The exception to this is in the north end of the mine where the ore contains more pyrite than it does in the other parts of the mine. Whenever there is any question as to the grade of the ore, special samples are taken under the direction of the engineering department. If the ore in question is in a stope, channel samples are cut, usually at 5-foot intervals, and an assay plan is made of the stope.

An ore reserve estimate is made every three months. As the ore is of comparatively constant grade, tonnage alone is considered in making this estimate. Three general rules governing the calculation of reserve ore are as follows:

1. A stope silled-out, first cut taken, but having no raise run in ore connecting the stope with the level above, is considered as having no reserve.

2. A stope silled-out, first cut taken, and having a raise run in ore connecting the stope with the level above, but with no ore silled-out above this level, is considered as having a reserve. In calculating this reserve the area of the stope is considered as the base of a cone and the height to the next level as the height of the cone. The reserve is calculated as a cone, the area of the base times one-third the height.

3. In calculating the reserve of a stope silled-out and connected by a raise or raises in ore to the level above, with a stope being silled out on this level above, the average area of the two stopes and the distance between them is used.

A new plan is made of each stope after each cut is taken, and the latest plan is used in calculating the reserve of any stope. The tonnage of ore reserve is calculated on the basis that 11.62 cubic feet of ore in place is equal to 1 short ton. Possible ore and probable ore do not enter into the reserve calculations. Over a period of years the recovery of the estimated ore has been over 90 per cent. The other 10 per cent includes ore lost by caving, ore of too low grade to mine or pinching out, and overestimation. The factor, 11.62 cubic feet per ton of ore, takes into consideration this 10 per cent discrepancy and compensates for it.

METHODS OF DEVELOPMENT AND MINING

The mine, operated by means of one vertical shaft of three compartments, is producing from 13 levels at the rate of 30,000 tons per month. Ore is broken and hoisted on two shifts, while on the third shift supplies and timber are lowered and some ore is hoisted. The shaft, which is timbered, has two hoisting compartments and a manway. The ore is hoisted in 3 1/2-ton skips hung below double-decked cages. The cages are not removed during hoisting operations. A second shaft has been started to handle the ore from the lower levels, as the ore bodies

are some distance from the shaft on the present lower levels. The new shaft has four compartments. There are two hoisting compartments, a chippy or man-hoist compartment, and a manway. This shaft will have steel sets instead of timber and, aside from the collar where concrete was used, it will be lined with native hardwood.

The sixth level, which is the first producing level, is 160 feet below the collar of the shaft. With three exceptions, the levels are 100 feet apart. However, after watching and checking up on two levels that are 130 feet apart, it was decided to space future levels at 150 feet. Aside from the extra hazard in driving the long raises to the level above, no difficulty is expected with levels at 150-foot intervals. The development plan of the company calls for sinking the shaft and starting a new level each year.

The general development plan for any one level is as follows: The ore bodies are projected from the level above so that their approximate location is known. Figure 2 shows the development on the 1,300-foot level. After the station has been cut, the main haulageway is driven towards the first ore body. When within 50 or 60 feet of the ore body, a crosscut is started to cut the ore. As the ore bodies lie more or less on a straight line, especially on the lower levels, the main haulageway is in reality a drift. This drift, in the footwall of the ore bodies, extends from the shaft to the farthest ore body. After the crosscuts from the main drift have cut the ore body, drifts are started to follow the ore, with crosscuts at suitable intervals for chutes and manways. On each level one or more raises to serve as manways and a raise or two to serve as ore passes are run to the level above.

DEVELOPMENT DETAILS

Shafts.- The compartments of No. 1 shaft, the present operating shaft, are each 4 feet 8 inches by 6 feet in the clear. The timber used is 8 by 8 inch hardwood which, although extremely heavy and hard to handle, makes excellent shaft timber as it lasts indefinitely, especially when wet. Shaft sinking is done by a local crew of natives. All drilling is done with 48-pound jackhammers. The standard shaft round consists of 22 to 28 holes, 5-1/2 feet deep. Each hole is loaded with five to seven sticks of 40 per cent gelatin dynamite and detonated by means of No. 8 electric, 1 to 10 delay, caps connected to a 110-volt light line. A sinking hoist, placed on the last operating level, handles the men, supplies, and muck for the shaft. Guides are placed in the manway compartment of the shaft below the level of the hoist, and a door to dump the bucket is placed at the station where the hoist is operating. The muck is shoveled into an 18-cubic-foot sinking bucket by shaft muckers using short-handled shovels exclusively. The bucket is hoisted to the level of the hoist and dumped into a car. When about 15 cars have been loaded, they are taken to some level above where the muck can be dumped into a stope for fill.

The new or No. 2 shaft has three compartments, 4 feet 8 inches by 5 feet in the clear, and one compartment 5 feet 6 inches by 5 feet in the clear. The larger compartment will be the manway compartment and will carry all electric cables, air and water pipe, as well as staggered ladders. A contract has been let to sink this shaft 1,800 feet.

Drifts and Crosscuts.- All drifts and crosscuts are nominally 5 by 7 feet. Track is carried on a .5 per cent grade. All drifting and crosscutting is done on contract, and about 10 per cent of the drifts and crosscuts have to be timbered. Timber is placed by men furnished by the company to the contractor. The contractor is paid for the timber placed and is charged for the timbermen furnished to him.

All drifts and crosscuts are driven with medium weight, Leyner-type drifters using 1 1/8-inch hollow round steel with a standard 5° and 15° cross-bit the machine working on a column and cross arm. A drift or crosscut crew consists of four men. The machineman and helper drill the round and blast on one shift, and two men on the other shift clean out the muck. This plan works out in some drifts, but in others three shifts are required to drill, shoot, and muck a round. Such delays are generally incurred in disposal of waste. The miners are generally good workers, but most of them must be taught to run a machine, as they usually have never seen one before working at this mine. The jackhammer is in high favor with the few miners who have worked in Spain, who if not watched will use this machine exclusively. At the slightest indication that something is wrong with the drifter, the miner will take down the bar and drill with a jackhammer. Education of the local miner progresses slowly.

A standar "wedge-cut" round is used with 15 to 18 holes, as hown in Figure 3. Holes are loaded with five sticks, and the cuts have 6 sticks of 40 per cent gelatin dynamite. This round has been found best adapted to this ground; there is some difficulty, however, in getting the miners to point the holes as they should.

The miner must use his own judgment in blasting, as one round may be in shale, the next few rounds in quartzite streaked with quartz stringers, and then the next back in shale. The average round is drilled to break about 4 feet, although there are a few of the better miners who consistently pull 5 feet. No trouble is experienced with the steel, because of the excellent blacksmithing on the surface. There are no "tool nippers," each miner bringing in and taking out his own steel. Occasionally some ambitious miner will have a good supply stored away for a rainy day in some out-of-the-way place. It is probable that eventually the steel will be checked out to the miner as are the machines and tools.

Raises-Standard raises are 5 by 5 feet in cross section. All raises are driven with hand-rotated (dry) stopers using seven-eighths-inch square solid steel. A raise crew consists of one miner and a helper on each shift. A "wedge-cut" round is used, drilled to break 3-1/2 to 4 feet. All stulls and staging are brought from the station and are placed by the miner and his helper. As the majority of the raises are fill raises from stopes, requiring no tramming of muck between rounds, good progess is made in all raises. Only about 1 per cent of the raises require timbering. If necessary to timber, 8-inch round cribbing is used, which is also placed by the miner and his helper. In most raises all material is pulled to the face by rope. A rope and the stulls provide the only means of getting up or down until the raise is holed through.

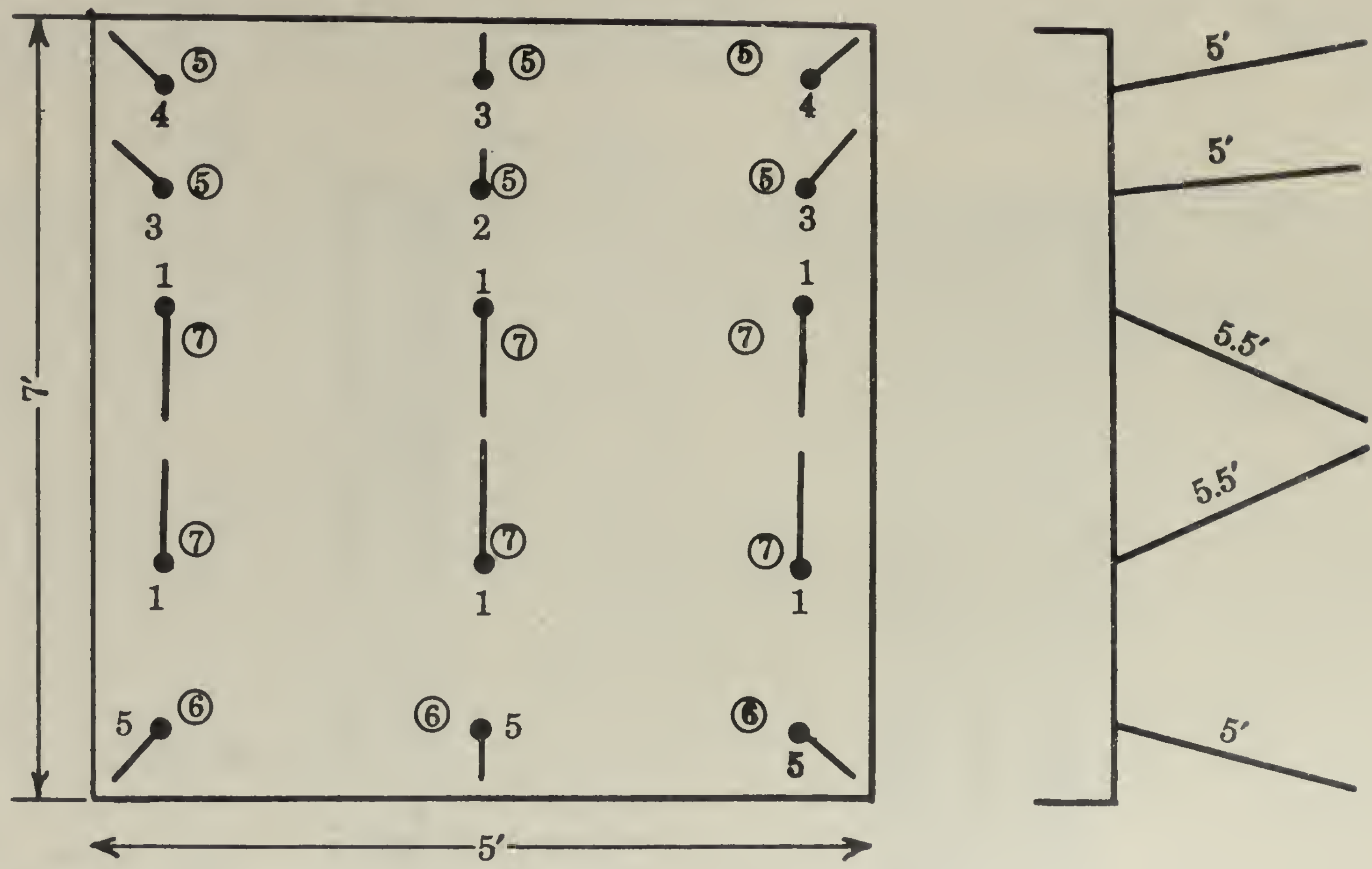

Figure 3.– Drift round.
(From "Drilling and blasting in some American metal mines," by Theodore Marvin.)
Not drawn to scale

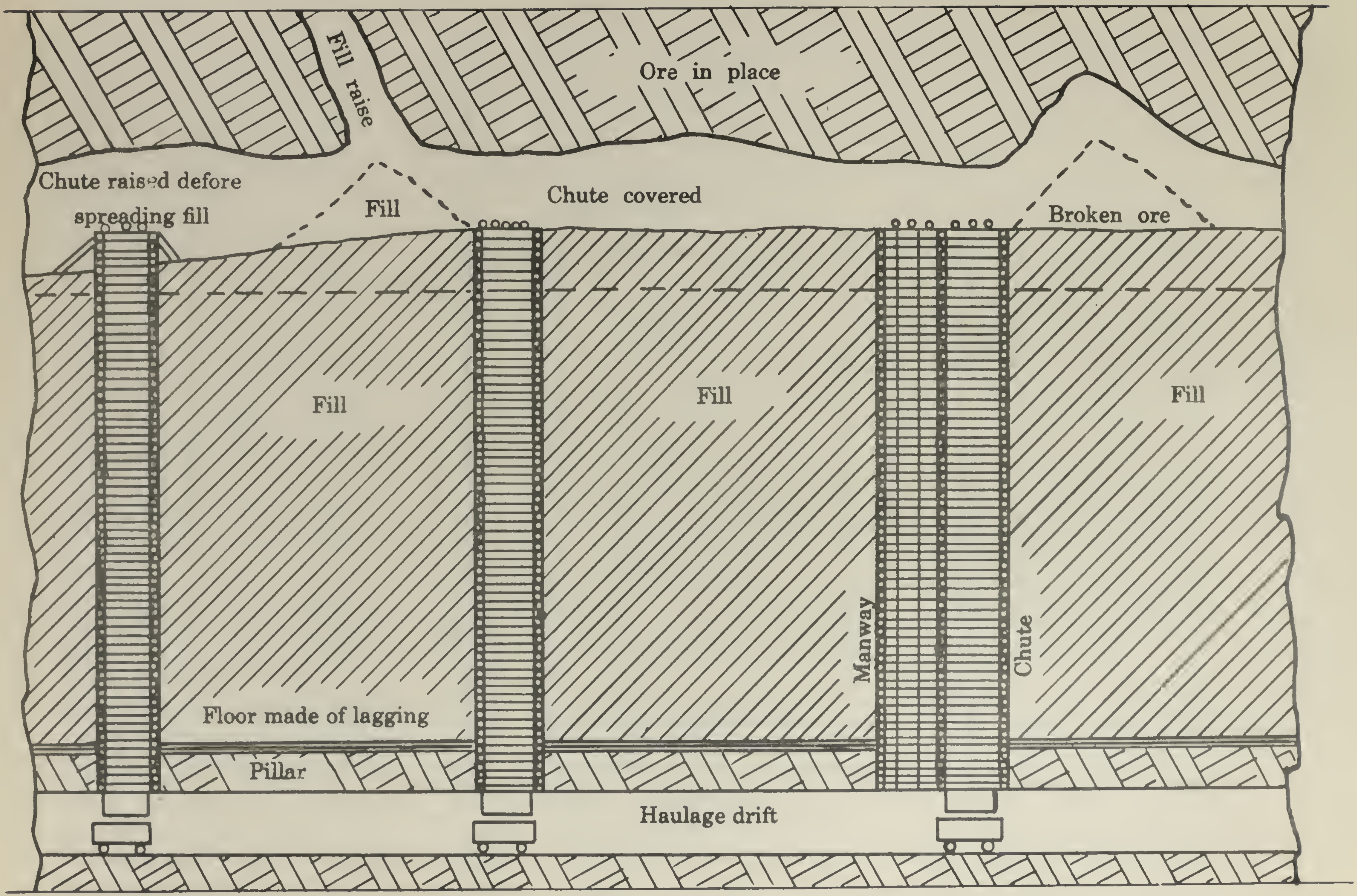

Figure 4.– Longitudinal section through stope, showing cut-and-fill mining. Not drawn to scale

STOPES

After an ore body has been located, a drift is run on ore the length of the ore body. Chutes are spotted at 50 foot intervals, and every third chute has a manway alongside. The chute raises are carried up about 20 feet and silling-out is started. The first floor of the stope is 14 feet above the floor of the drift below. Chutes are constructed of 10-inch round timber with 3-inch pine floor and sides. Five rings of cribbing are placed in each chute below the floor of the stope to give a base to build from, as the stope is taken up and the chutes are raised with each cut. The cribbing used is 8-inch round pine framed on the surface. The cribbing is built up without the use of nails, the tongues holding the pieces in place until fill is packed around the cribbed-up raise. Arc-shaped, iron chute doors, fabricated in Havana, are used to control the flow of the ore from the chute. As the largest pieces of ore coming from a chute are not larger than 12 or 14 inches, the doors have proved a great saver of time. Figure 4 is a longitudinal section through a cut-and-fill stope.

In silling-out a stope the hanging wall and footwall of the ore body are sought first, as soon as there is enough room to work. When the walls are found, the stope is silled-out from end to end. This first, or sill floor cut, is usually about 12 feet in height; all stope backs are flat. If the stope sills-out wide, crosscuts have to be driven from the drift below to allow more raises to be brought up to the stope for ore chutes. On the first cut all stringers and showings of possible ore are followed to make sure that all ore has been silled out. As there are horsts of waste in some of the ore bodies, and as the ore bodies are cut by slips and fractures which have caused little or no displacement, it is very easy to mistake the walls of the slips for ends of the ore body. Consequently the first cut may take in ore that is too poor to mine as the stope goes up. As the largest ore reserve estimate is made from the first floor, the estimate will be too large unless the silling-out of the stope is carefully watched.

All ore mined in the stopes is drilled with hand-rotated dry stopers and 48-pound jackhammers, both using seven-eighths-inch solid square steel. The holes are loaded with five sticks of 30 per cent gelatin dynamite and are fired by means of a 7-foot fuse and No. 8 cap.

A cut is started in the center or end of the stope, depending on its size. The cut is started with a stoper and carried through with jackhammers. Most of the ore is then slabbed off by flat holes drilled with jackhammers. The number, depth, and placing of the holes depend on where the round is being drilled, whether in the center or side of the stope, and the nature of the rock being drilled. An average round breaks from 8 tons along the sides to 15 tons when the round is in the center of the stope and the back is good. All holes are blasted by the miner when going off shift or at lunch time. All broken ore is placed in the ore passes to the level below by shoveling direct into the raise or by wheelbarrows. An effort is made to keep the chutes close enough together so that the ore can be mucked directly into the raise, but sometimes after a stope is halfway up to the level above it widens out or becomes a little longer and necessitates

the use of wheelbarrows. The ore is hard and breaks comparatively small. All pieces over 12 inches are broken either by hammer or bulldozing before they are put into the raise.

Four-hour carbide hand lamps are used in the stopes; they are used everywhere in the mine.

While the stope is being silled-out, fill raises are run from the stope to the level above. Formerly when surface fill (brought from centrally located glory holes on the surface to the different stopes by means of raises), as well as development waste, was put in the stopes, it was the practice to drive the fill raises at 80-foot intervals. During the past year and a half the surface fill has been dispensed with and de-slimed mill tailing used in its place.[3] Due to the ease in handling the tailings, known locally as "sand fill," fill raises at 100 to 150-foot centers are sufficient. At least one raise to the level above is equipped with ladders, and in the larger stopes two or three raises are so equipped. All cribbing and timber for use in the stope are thrown down a fill raise from the level above. Machines, drill steel, and other supplies are taken in and out of the stopes by the stope crew.

With one exception, no stopes have required square-sets on the first or sill floor. Ordinarily the only timber used in the stopes for 70 or 80 feet above the level is the cribbing for the ore passes and manways and the plank lining for the ore passes. However, in a number of ore bodies the ore has been so fractured that these stopes require square-set timber. After a stope has been carried up to within 15 or 20 feet of the level above, square-sets are always used. All sets are filled as soon as a cut is completed. This permits carrying the stopes up and past the level and removing the ore from the level pillar of the stope above. While the removal of the level pillars from below the smaller stopes causes no trouble, as a rule the removal of a pillar from beneath a large worked-out stope causes enough subsidence to prohibit efficient tramming through the old drifts below the stope. If there is no haulage drift in the footwall of the stope, one is then driven before the pillar is removed. During 1928 approximately 30 per cent of the total mine ore came from square-set stopes.

When a stope has been completely silled-out, the first cut taken, and a floor of hardwood slabs and poles laid down, it is ready for fill. On an average, after a cut has been taken the back is 12 feet above the floor, and 6 or 7 feet of fill is placed in the stope. Before the fill is put into the stope the ore passes to the level below are cribbed up to the required height. If development waste is used, it is dumped down the fill raises and spread in the stope either by shovel, shovel and wheelbarrow, or by scraper. Part of the contract crew spreads the fill; the contractor is paid 20 cents for each ton of fill spread by hand and 5 cents per ton for fill spread by scraper. The scrapers are operated by means of double-drum tugger hoists. Due to the care the hoist must have, the necessity of changing the hoist to keep it lined with the cable, etc., the scrapers have not proved popular with the men.

3 Richert, G.L., Filling Stopes with Mill Tailings. Eng. and Min. Jour., Mar. 2, 1929, p. 348.

At present one-half the fill tonnage is made up of sand fill. The amount of sand fill used depends entirely on the amount of development waste that must be disposed of as fill.

Briefly, the sand-fill system consists of pumping mill tailings to a 30-foot bowl classifier situated on the surface near a raise leading into the mine. Figure 5 shows diagramatically the Matahambre system of filling stopes with mill tailings. The slime overflow from the classifier is sent to the tailings pond, and the rake product drops into a hopper. Water is added to the hopper and the course tailings are washed into a 2 1/2 inch special rubber-lined pipe. This pipe is strung through a line of raises from the surface to the 1,200 level. The sand fill is taken through the pipe to the stope to be filled. Before the stope is filled, the cribbed up raises and manways are wrapped with a commercial grade of burlap to prevent the sand from washing out between the cribbing. By means of pipe or hose the sand is directed wherever wanted. By building up small sand dams it is possible to fill any part of the stope. After 12 hours the water has seeped out through the burlap into the ore passes, and the floor of the stope is almost as hard as ordinary soil. After two years of experimenting it was found that rubber-lined pipe was best for handling this fill. Standard pipe, extra heavy pipe, and special rubber hose were tried, but the rubber-lined pipe proved to be the best. The advantages of this filling system are ease of handling, no spreading, a sufficient amount of fill always available, the fact that the fill can be changed from a stope in one end of the mine to a stope in the other end of the mine by breaking and making a few pipe connections, and the fact that the sand enters all cracks and fissures and forms a solid pillar between the walls of the stope.

UNDERGROUND TRANSPORTATION

All ore hoisted to the surface is loaded into the skip from pockets on the 800 to 1,200 or 1,500 levels, or by dumping cars directly into the skip on the 1,600 and 1,700 foot levels. The installation of an ore pocket below the 1,800 level is 75 per cent complete. When this is finished the ore from the 1,600, 1,700 and 1,800 levels will be handled through this pocket. On the 600, 700, 800, 900, 1,000, and 1,100 levels, the ore from the north part of the mine is sent through ore passes to the 1,200 level, where it is taken to the station and pocket by motor train. The ore near the shaft and the south part of these levels is trammed by hand to the ore passes or pockets at stations. All cars are 20-cubic-foot capacity, end-dump, and have roller-bearing wheels. On the 1,200, 1,300, 1,400 and 1,500 levels the ore is taken to the station by small storage-battery "trammers" or locomotives pulling from six to eight cars. As the grade is with the load, the length of the train is not dependent on the loaded cars that can be handled, but on the number of empties that can be taken back against the grade. There are six of these motors underground and one operating on the surface hauling supplies and timber from the tram terminal. The average duty of the motor is 8 hours; a crew of two men is required, and an average trip of 1,200 feet is made, hauling 100 to 150 cars of ore per shift. The brakeman assists in loading, or loads the cars from the chutes below the stopes. All motors are brought to the surface at the end of each shift, the batteries are changed, and the motors are sent back for the next shift. Only on special occasions are motors worked on three shifts.

The mine is 90 per cent equipped with arc-shaped iron chute doors. Those used in the chutes below the stopes are 30 inches long, whereas those used on chutes in ore passes between levels or leading to the skip pockets at the stations are 40 inches long.

On the levels above the 1,700 level, the track is 18-inch gauge and 18-pound rails are used. Below the 1,700 level the track is 18-inch gauge and 24-pound rails are used. Grade is carried at plus one-half of 1 per cent going from the shaft.

PERCENTAGE OF EXTRACTION

The limiting factor in the percentage of extraction in square-set stopes at Minas de Matahambre is the price of copper. In straight cut-and-fill stoping practically all mineral-bearing rock is removed. When square-sets are used, the cost of timbering makes it necessary to leave some of the low-grade material which is not worth extracting. For this same reason it is sometimes not economical to extract low-grade portions of pillars. As stated previously, on the first or sill floor every effort is made to get all the ore in sight to make sure that the stope has been completely silled-out. Consequently, when a stope comes up under a pillar and square-sets are used, in many cases not all the ore that was silled-out above the pillar will pay to square-set. However, as over 90 per cent of the estimated ore (from ore reserve estimates) is recovered,- the other 10 per cent includes overestimation, and loss by caves and all other causes - the amount of ore left or lost is small.

No waste is sorted on the surface, although high-grade ore is sorted on a picking belt. However, sorting of waste is carried on underground in all the stopes. This applies especially to stopes with ore below average grade which are being mined with the expectation that they will eventually improve and come up to grade. If the waste is sorted conscientiously and piled up where it can be measured, it is sampled, measured, and paid for at 5 cents per ton more than the miner received for his ore. Before this practice was instituted, it was thought that the miners would shoot down everything in sight and pile it up for waste but it has not worked out that way. Car checkers count the cars and examine the ore at the station. The contractors are paid on a "per car of ore" basis, and any cars containing large proportions of waste are not paid for.

VENTILATION - NATURAL AND MECHANICAL

Before the installation of the fans the mine had a fairly efficient natural ventilation system.[4] Aside from raises to glory holes, two lines of raises had been run from the 1,300 level to the surface and one line from the 700 level to the surface. Two of these lines of raises are single compartment, 5 by 6 feet, and the other is a double compartment raise 5 by 10 feet. The single compartment raises were used as manways and for ventilation. The double compartment raise carried the sand-fill pipe line and was also used for ventilation. In addition to these raise connections to the surface, the north part of the Matahambre mine has been connected on nearly all levels to the Ruiseñor, a north extension of the Matahambre.

4 Richert, G. L., Mechanical Ventilation at the Minas de Matahambre. Min. Cong. Jour. July, 1928, p. 491.

The Ruiseñor levels are connected to the surface by a double compartment raise which is naturally downcast. This natural ventilation makes the main shaft and the Ruiseñor raise downcast, while the three raise lines to the surface are upcast. This condition has not been changed by the addition of the fans. Although the natural ventilation removed the gas and smoke after blasting, the heat and humidity in practically all parts of the mine was oppressive.

In April, 1927, Mr. Charles A. Mitke made a study of the mine ventilation and found that temperature and humidity conditions were as follows:

Temperatures underground ranged from 71 to 93° and in the majority of places were over 80°. There was almost complete saturation of the air in the mine due to water in the shaft and in the drifts. Air with surface humidity of 46 per cent entered the 1,200 level 96 per cent saturated. At Mr. Mitke's suggestion three fans, each of 45,000-cubic-foot capacity, driven by 50 hp. motors and operating against a minimum of 3 1/2-inch water gauge, as suction fans were installed on the surface over three of the raise lines.

In addition to the fans the necessary bulkheads, doors, and runarounds, were installed to direct the air properly through the mine. This arrangement increases about six times the amount of air, approximately 21,000 cubic feet per minute, which entered the mine by natural ventilation. It was estimated that the mine shaft, as an intake, would supply 72,000 cubic feet of air per minute, the Ruiseñor raise connection 6,000 cubic feet per minute, and other raises with surface connections would supply the remaining 27,000 cubic feet, while the three fans with a capacity of 45,000 cubic feet each would exhaust 135,000 cubic feet per minute.

While temperatures have not been materially reduced, the increased volume and velocity of the air through the mine has (1) bettered the morale and spirits of those employed underground; (2) speeded up the work, especially tramming; decreased the rate of decay of timber, especially in the main haulageways to the shaft, for the fungus and "cotton" have practically disappeared.

WAGE, CONTRACT, AND BONUS SYSTEM

All labor at Minas de Matahambre, with the exception of the staff, is hired at the mine. Contrary to general opinion, there are very few Cubans employed underground. The Cubans are by nature farmers and do not care to work underground. The mine labor is made up of Spaniards, Russians, and men of other central and southern European nationalities; there are at least 27 nationalities represented on the payroll. The Spaniards, many of whom have worked in the mines in Spain, are good miners and make good contractors and bosses. Some of the Russians have also had mining experience and make good contractors and timbermen. However, probably 60 per cent of the underground labor have not had previous underground experience. This causes a shortage of good machinemen, as most of these men must be taught to drill. The others readily adapt themselves to mucking and tramming. All in all, the underground labor at Matahambre is much better than the general run of mine labor in tropical countries.

Company wages are $3 for timbermen, $2.50 for miners, $2.25 for motormen and timbermen's helpers, and $2 for muckers and trammers. All work underground is done by contract, and all contractors are guaranteed company wages.

A stope contractor has charge of the stope where he is working, or in larger stopes where there are two or three contractors, he has charge of and is responsible for only a certain part. The contractor is paid a fixed price for all work done and is charged for fuse, caps, dynamite, carbide, and his men's time. The men working for the contractor are paid by the company at company wages Average prices paid in stopes are:

	Price
Ore per car	$0.60 to .70
Spreading fill	.20 to .30
Square-set timber, per piece	.75
Cribbing, per piece	.15
Slabs laid on the first floor before fill is placed in the stope	.10
Lagging and planks	.10
Hand-sorted waste (per ton)	.75
Chutes, single	12.00
Chutes, double	15.00

The company furnishes the timbermen to put in the chutes, and the contractor is charged with their time.

All timber and material must be taken from the shaft station to the stope by the contractor's men. Each contractor has an "associo" who is in charge of the other shift, and who shares equally with the contractor in the money.

On drift contracts there are four men, the contractor and his "associo" - both are miners - and two muckers or helpers. The average price paid for drifts is from $12. to $18 per meter, depending on the nature of the ground. The contractor is charged with helper's wages, as well as with the dynamite, fuse, caps, and carbide.

Raise contracts have four men - two miners and two helpers. The average price paid for raises is $12 per meter for a single raise in ore, 75 cents for each car of ore, and $15 per meter for a single raise in waste. Special raises for ore passes, and those requiring timbering are at special prices ranging from $15 to $22 per meter. It was found that unless a raise man was paid for ore, when he was driving in ore, he turned the ore in at the station for some stope contractor. Under the present system his raise price drops when he is in ore and he must turn his ore in under his own name to receive pay for it.

All work is measured by the engineering department twice a month, on the 15th and 30th. The men are paid only once a month, although they may draw part of their money each Saturday. The measurement on the 15th is carried through the same as at the end of the month. All money earned and all charges are put on the contract statements. This gives the mine superintendent a chance to see how the

work is going, and who is and who is not making money. He then has a chance to find out the reason and to try to remedy matters. As the majority of the labor can neither read nor write, the thumbprint made with ink from a stamp pad goes as a signature. Everything on the contract sheets must be as simple as possible, for these men are dependent on those who can read for an explanation of their contract statements.

A speed bonus is in effect for drifting and crosscutting. Any contractor who drives 70 feet of drift in one month receives $1 extra for each meter driven, for 85 feet $2, for 100 feet $3, and for 115 feet $4 extra. No one has made the 115-foot mark as yet, although a number have made more than 100 feet. A raise bonus in force adds $3 per meter to the price of a raise after it is 50 feet vertically above the starting point. From July, 1927, until October, 1928, a general bonus system was in effect in which all participated. Under this system a stope contractor and his "associo" were allowed to make $3 per shift worked before paying a bonus to the men. If more than $3 per shift was made, 40 per cent of the money over $3 was divided between the contractor and his "associo" and 60 per cent between the men working for him. In drifts and raises the base pay was $2.50 per shift. If more than $2.50 was made, 60 per cent of the amount over $2.50 was divided between the contractor and the "associo" and the remaining 40 per cent among the men according to the shifts each had worked. This system was not entirely satisfactory; at the present time after the contractor has earned his maximum wage allowance before bonus distribution, both contractor and men participate equally in the bonus, in proportion to the number of shifts each man has worked.

FIRE HAZARDS

The fire hazards underground are few, but they are carefully guarded against. The main shaft, which is downcast, is comparatively dry from the surface to the 700 level, a distance of 244 feet. Below the 700 level the shaft is wet and not likely to burn. Due to the high humidity in the mine all timber in the drifts is damp and would be very hard to set afire. However, to eliminate as much as possible any likelihood of the occurrence of fire underground, no rubbish or trash is allowed to accumulate in the mine. As a means of handling a fire, if one should start, 50-foot reels of standard fire hose connected to a 2 1/2-inch pipe line are placed at the collar of the shaft and on the 600 and 700 level stations. The pipe line is connected to a fire pump that is hooked up to draw from an 8,000-gallon storage tank or from the main camp reservoir supply line. A fire line, 2 1/2-inch pipe, with hose boxes and hose at suitable intervals, has also been placed on the surface to protect the surface buildings and the timber yard. In addition to the hose on the 600 and 700 levels, chemical fire extinguishers have been placed in all powder magazines and at the pump stations on the 900, 1,300, and 1,700 stations. Because the shaft is naturally downcast, wooden doors faced with metal, having thei frames set in concrete, have been placed on all levels as near the shaft as possible; the average distance from the shaft is 150 feet. If there is fire in the shaft, these doors can be shut and so protect the levels from smoke and gas. As a warning signal to the men underground in case of fire, there is also on the surface a Mercaptan tank and feeder. The feeder is connected to the main air line going down the shaft. A standard set of fire rules have been drawn up and posted

in conspicuous places. These rules give specific directions as to the handling of the fire fighting equipment on the surface as well as underground, depending on the location of the fire, where to go and what to do, when to shut down fans, and whom to notify first. As there are a telephone exchange and two timekeepers in the mine office, 50 feet from the shaft, most of the first work would fall to them. The company has recently purchased six McCaa two-hour breathing apparatus. As soon as possible a crew will be trained to use this apparatus under direction of men trained by the Bureau of Mines.

SAFETY METHODS AND FIRST AID ORGANIZATIONS

An active safety first organization has been in operation at Minas de Matahambre since the latter part of 1927. The organization consists of a general safety committee and several local committees. All committees meet every other week. The general manager acts as the chairman of the general committee and the safety engineer acts as the secretary. The members of the general committee are the heads of the different departments and their assistants. The chairman of each local committee is the head of the department represented, and the members are chosen from the men working in that department. The members of the local committees are paid $1 for each meeting attended. The members of these committees are changed every three months. Local committees have been organized in the mine, mill, surface, machine shop, power plant, and dock departments. The business of the general committee is to discuss safety-first problems. The business of the local committees is the actual instruction of the men on the job in the matters of safety and first aid. To help keep up the interest in this work the company has supplied "committee" badges for the members of the local committees, pins for one year's service without accident, instruction and rule books which are given to each man when he goes to work, a "safety flag" which is flown over the department having the smallest number of accidents for one month, a glass-enclosed and electrically lighted bulletin board, with signs and posters, and a large clock with one hand, the hand pointing to the number of days the mine has worked without an accident. In addition a bonus of $15 is given each shift boss in the mine having a record of 1,000 man-shifts without a lost-time accident and a bonus of $25 is given for each succeeding 1,000 shifts without an accident. There are special prizes consisting of a $125 watch for the mine-shift boss having the least number of accidents during one year, $50 for the second best record, and $25 for the third best record.

Some changes and improvements that have been made as a result of the activities of the safety first organization are:

1. A reduction of about 60 per cent in the number of accidents.

2. The use of leather shoes instead of canvas slippers by all men working for the company.

3. The installation of first-aid boxes on all levels of the mine. Each box contains bandages, gauze, cotton, and a bottle of mercurocrome. There is also a Stokes wire stretcher on all stations in the mine and in all plants on the surface.

4. Maintenance of the fire prevention and fire fighting system and apparatus previously described.

5. The provision of electric lamps for use in magazines in case the power goes off.

6. The use of "hard-boiled hats."

7. The use of screen goggles by all men breaking rock.

8. The use of separate canvas sacks for carrying dynamite and capped fuse to the working places.

9. The covering of all unused raises in the stopes and of raises and ore passes in use with grizzlies made of steel rails.

10. The changing of all open electric switches on the surface and underground for closed types of switches.

11. The guarding of all machinery on the surface as well as in the mine by use of guards over gears, etc.

12. A general "safety first - watch your step" spirit or attitude in all employees.

TABLE 1 - SUMMARY OF COSTS

Name of mine: Minas de Matahambre, S. A. Period covered: Jan. - Dec., 1928.

Tons of ore hoisted during period: 364,746 short tons

Mining method: Cut-and-fill and square-set.

Underground costs per ton of ore hoisted

	1 Labor	2 Super-vision	3 Compressed-air drills and steel	4 Power cost	5 Explos-ives	6 Timber	7 Other Supplies	8 Total
Development: In rock	$0.334	$0.060	$0.067	$0.009	$0.105	$0.077	$0.059	$0.711
Mining	.559	.095	.085	.005	.095	.192	.059	1.090
Fill	.134	- -	- -	.003	- -	- -	.040	.177
Transportation (underground) ...	.281	- -	.005	.035	- -	- -	.067	.388
General underground expense ..	.106	- -	.004	.035	- -	- -	.039	.184
Total	1.414	.155	.161	.087	.200	.269	.264	2.550
Surface expense (directly applicable to underground operation)	.019	- -	- -	- -	- -	- -	- -	.019
Total	1.433	.155	.161	.087	.200	.269	.264	2.569

TABLE 2 - SUMMARY OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Name of mine: Minas de Matahambre, S. A. Period covered: Jan. - Dec., 1928.

Tons ore mined and hoisted: 364,746. Mining method: Cut-and-fill and square-set.

	Development	Mining (stoping)	Total
A. Labor (man hours per ton):			
Breaking (drilling and blasting) ..	0.38	0.69	1.07
Timbering and filling	0.11	0.67	0.78
Mucking	0.51	0.92	1.43
Haulage and hoisting	- -	0.64	0.64
Supervision	- -	- -	0.11
General	- -	- -	0.28
Total	1.00	2.92	4.31
Average tons per man per shift	8.00	2.74	1.86
Labor, percentage of total cost ..	19.91	39.87	59.78
Av. tons per man-shift on surface properly chargeable to underground operation	- -	- -	211.71
B. Power and supplies per ton:			
Explosives (lbs. per ton)	.58	.44	1.02
(kind and grade) gelatine	- -	10%-30%-40%	- -
Timber (framed, b.m.)	.344	.802	1.146
(props, pcs.)	.023	.125	.148
Power (kw.h. per ton)			
Air compression	3.50	4.50	8.0
Hoisting	- -	3.00	3.0
Pumping	1.03	.76	1.79
Ventilation	- -	1.22	1.22
Total	4.53	9.48	14.01
Other supplies in percentage of total supplies and power	11.27	16.13	27.40
Supplies and power, percentage of total cost	16.05	24.17	40.22
C. Percentage of total cost	35.96	64.04	100.00

Note: Figures for development and mining are both based on total tonnage produced rather than on ore produced from development and mining respectively.

TABLE 3 - DETAILS OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

	Sinking	Drifting	Cross cutting	Raising
Size of excavation	17 x 8 ft	5 x 7 ft	5 x 7 ft	5 x 5 ft
Timbered or not	Yes	10%	10%	1%
Physical properties of rock	Hard and firm in quartzite Soft and loose in shale			

	Total development
A. Labor (man hours per foot):	
Breaking (drilling and blasting)	3.26
Timbering	1.38
Mucking	6.61
Other labor	1.00
Total labor	12.25
Feet per 8-hour shift	0.65
B. Power and supplies (per foot):	
Explosives (lbs. per ft.)	7.48
Timber (per ft.)	$1.07
Total power (kw.h.):	
Air compression	45.4
Hoisting	- -
Haulage	- -
Ventilation	15.9
Other supplies	$1.17
C. Labor (percentage of total cost):	55.3
Power and supplies (percentage of total cost)	44.64

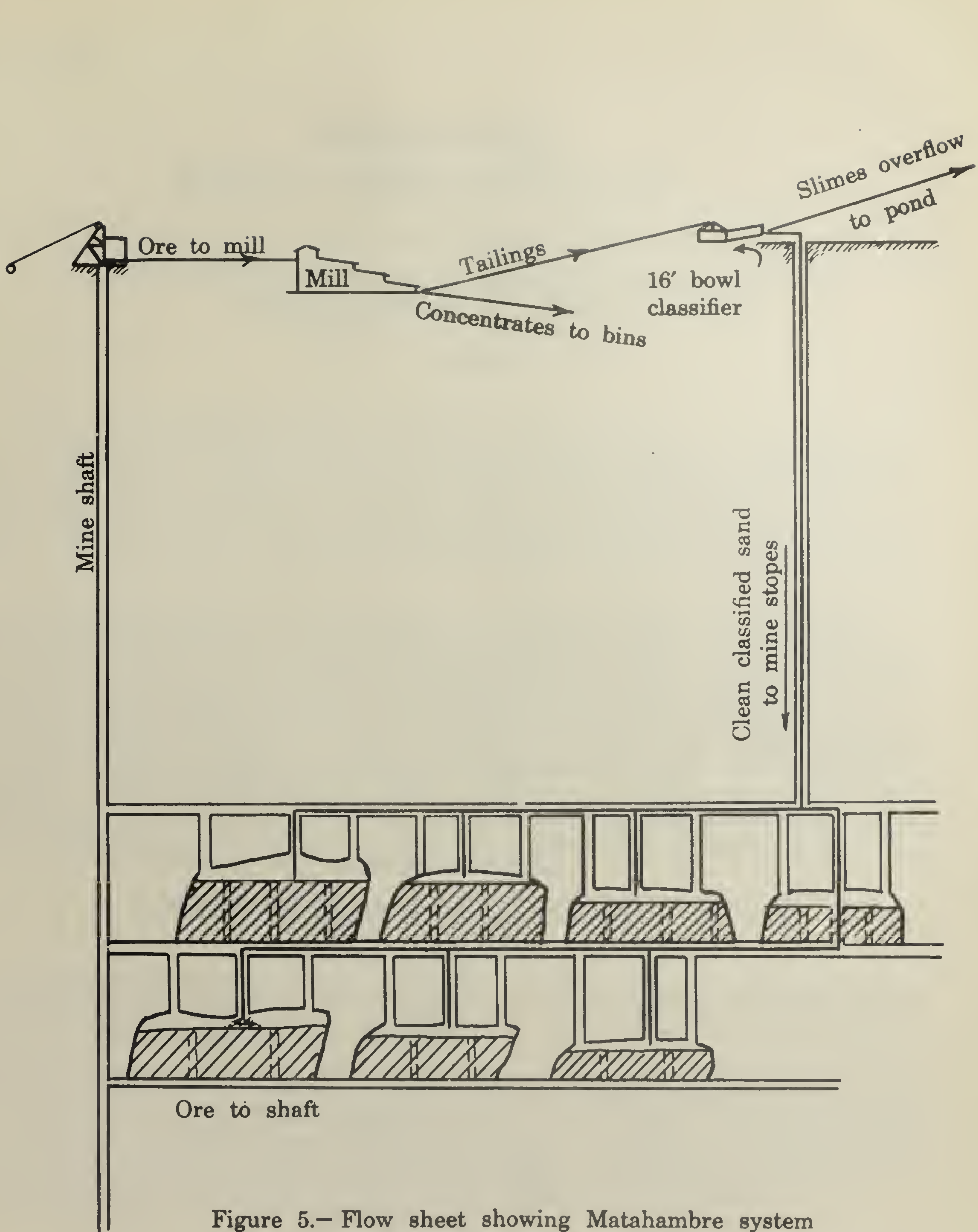

Figure 5.– Flow sheet showing Matahambre system of filling stopes with mill tailings

Circular 6146.
June, 1929.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SAFEGUARDING ELECTRICAL EQUIPMENT USED IN GASSY MINES [1]
EUROPEAN PRACTICE: IV - FRANCE

By L. C. Ilsley[2]

Cooperation between the United States Bureau of Mines and the Safety in Mines Research Board of Great Britain, continuous since 1924, has made possible this and other papers on safety subjects. Grateful acknowledgment is made to representatives of the board for their assistance in arranging visits to several mine-safety stations, and to F. R. Wynne, Deputy Chief Inspector of Mines, for arranging visits to mines in Great Britain and certain other countries in Europe.

During the summer of 1927 the writer had the privilege of visiting the mine-safety testing stations in Great Britain, Belgium, Germany, and France, in the order named. All of these countries have large coal mines, many of which are rated as gassy. Therefore, when the installation of electrical equipment is contemplated, each of these countries is confronted with the same safety problem as is the United States - that is, the development of electrical equipment that will not ignite gassy atmospheres should they through neglect or accident surround the equipment. The means employed by these countries in safeguarding gassy mines should therefore be of general interest to safety engineers of American coal mines, and a brief survey is given for each of the four countries mentioned. The fourth or last of these surveys covers conditions in France.

REGULATIONS COVERING THE USE OF ELECTRICITY IN FRENCH COAL MINES

The following regulations covering the use of electricity in French mines have been abstracted from the Decree of August 13, 1911, which it is understood is the latest decree bearing on the subject.

1 The Bureau of Mines will welcome reprinting of this article, but requests that the following footnote acknowledgment be used: "This paper represents work done under a cooperative agreement between the U. S. Bureau of Mines and the Safety in Mines Research Board of Great Britain. Printed by permission of the Director, U. S. Bureau of Mines. (Not subject to copyright.)"
2 Electrical engineer, U. S. Bureau of Mines.

Gassy Mines:

Article 134

....When portable electric lamps are used, a flame safety lamp is provided for each working place.

Article 216

The use of electricity is forbidden in mines liable to instantaneous outbursts of firedamp, except for portable electric lamps and shot-firing.

In other gassy mines, electrical installations can be made only in the intake air shafts, in landings of these shafts, and in drifts which receive air coming directly from the shaft without first having circulated in any of the working places of the seam, as well as in the immediate vicinity of these landings or drifts.

Armored cables can be placed in the return air of slightly gassy mines with the permission of the district mine authorities.

Article 217

In gassy mines, only blasting machines of a type approved by the Minister of Public Works can be used.

The blasting machines must be sturdily built and maintained in good condition at all times.

Article 218

Except for the rulings of Article 217, it is possible to make use of electric signals or telephones under the following conditions in all parts of gassy mines where tests of the atmosphere, made at least once a day, do not indicate a higher percentage of gas than 0.40

1. Permanent wiring should be installed in armored cable, flexible cables should be protected by flexible conduit.

2. The cables shall be placed as close as possible to the floor and protected against all causes of breakage.

3. Contact points shall be protected by a layer of oil at least 5 centimeters (1.97 inches) deep.

4. Apparatus capable of giving off sparks shall be enclosed in cases capable of resisting an internal explosion of fire damp. These cases must be constructed and maintained in such a way that the ignition can not propagate itself to the external atmosphere.

The use of signals must be stopped immediately if fire damp appears in greater amounts than 0.75 per cent in the vicinity of the installation or at any point along the lines between the installation and the intake air shaft.

Article 219

The installations must be maintained in a good state of insulation.

Insulations in contact with ground shall be checked at least every three months in case of permanent installations and one a month for temporary installations. Insulation between conductors of opposite polarity or different phase shall be checked at least every six months. The results of these checks shall be placed in a log book which shall be open to the mines department at all times.

Defects in insulation must be located and repaired as soon as detected.

Article 220

Temporary wiring and portable motors should be inspected at least once a week.

Nongassy Mines

Since electricity is practically prohibited in the working areas of gassy mines, it should be of interest to study some of the regulations issued in the same decree for nongassy coal mines. Some of the more important regulations follow:.

Article 25

Electrical installations should embody safety devices suitable to the highest voltage existing between the conductors and ground.

Electrical installations are divided into two classes, according to voltage.

First Class

A. Direct Current. - Installations in which the highest normal voltage between conductors and ground does not exceed 600 volts.

B. Alternating Current. - Installations in which the highest effective voltage between conductors and ground does not exceed 150 volts.

Second Class

Installations - d.c. or a.c. - having higher voltages than those mentioned above.

Article 26

Frameworks and driving mechanisms not forming parts of the current path, in machines belonging to electrical systems of the second class should be effectively grounded or effectively insulated from ground. In the latter event the machines shall be surrounded by a platform which

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[illegible]

[illegible]

[illegible]

[illegible]

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[illegible]

[illegible]

[illegible]

[illegible]

shall afford firm footing, shall be insulated from ground and be of sufficient width that it shall not be possible to touch the machine and any grounded body at the same time.

The ground or the electrical insulation must be maintained in good condition at all times.

The same rules hold for transformers serving installations of the second class; apparatus of this kind must be accessible only to those responsible for it.

Article 27

If a machine or an electrical apparatus of the second class is installed temporarily, the part of the premises affected by this machine or apparatus shall be made inaccessible to all but those in charge of it by a guard rail or equivalent device; a sign indicating the danger should be attached in a prominent place.

Article 29

Wiring on switchboards pertaining to systems of the first class should offer insultation and spacing of a character to avoid all danger.

Switchboards carrying apparatus and metallic parts of the second class should have platforms at the front (where the operating handles and instruments are) and be insulated electrically and built like the platforms surrounding machines.

When the metallic parts or apparatus of the second class are erected on the back of the board without protection, an unobstructed passage at least 1 meter (3.28 ft.) wide and 2 meters (6.56 ft.) high shall be maintained behind the aforesaid apparatus and metallic parts; access to this passage shall be by means of a door locked by a key; this door shall only be opened by order of the superintendent or by those authorized by him; entry to it is forbidden to everyone else.

Article 30

Passageways provided for access to unprotected machines and apparatus of the second class can not be less than 2 meters (6.56 ft.) high; their width, measured between the machines, conductors, or apparatus themselves as well as between them and the metallic parts of the building should not be less than 1 meter (3.28 ft.).

Conductors and apparatus of the second class, particularly on switchboards, should be clearly differentiated from others by some obvious marking - a coat of paint, for example.

In places where the floor or the walls are very conductive either by construction or by saline deposits, unprotected conductors or apparatus shall be rendered inaccessible by position.

Article 35

It is expressly forbidden to have work done on lines in the second class without having first disconnected both sides of the section to be repaired. The circuit shall be reestablished only by the express order of the superintendent; the latter shall have first been informed by each of the crew bosses that the work is finished and that the workmen have reassembled at a meeting place previously agreed upon.

All the time the work is going on the break in the line shall be maintained in a manner that the line can not be made alive except by order of the superintendent.

Under exceptional conditions, where public safety requires that work be done on charged lines in the second class, work shall not be started without the express instructions of the superintendent and with all the precautions he shall point out.

Article 37

Telephone, telegraph, or signal lines, particularly in mines having electric installations, affected by the working of the mine which are mounted for the whole distance or any part of it on the same supports as a power line in the second class, are subject to the rules governing installations of the second class.

Their dispatching point, their operating devices, or calling devices should be arranged in such a manner that it is not possible to use or operate them unless they are well insulated with respect to ground unless the devices are arranged in such a way as to insure the insulation of the operator from ground.

Article 38

The user is obliged to set up in a conspicuous place in rooms containing installations of the second class:

1. An official notice indicating "Danger" and expressly forbidding touching metallic parts or conductors subjected to voltage of the second class even with rubber gloves, or to work on these parts or conductors even with tools with insulated handles.

2. Instructions covering first aid to be given to victims of shock, drawn up in a way to conform to the terms which will be established by a decree of the Minister of Public Works.[3]

Article 39

In the two months to follow the promulgation of this regulation, the operator should send to the Chief Engineer of Mines a diagram of his electrical installations of the second class, showing the location of shops, substations, transformer stations, and the wiring.

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3 This decree has not been issued. It seems that one could substitute therefor the instruction appended to the circular of May 24, 1911, which we give hereafter (p. 181); the two tables it shows, will satisfy practically, if not legally, article 38 of the decree of 1911. (Recueil Des Textes Officiels Relatifs A L'Exploitation Des Mines De Combustibles. 1912 Edition.)

An accompanying statement shall indicate if, in accordance with the articles of this regulation in regard to machines and transformers of the second class, the frameworks and metallic parts, not part of the current path, are insulated electrically from ground or if they are grounded. The same statement shall give the necessary technical information to insure a countercheck on the execution of the provisions of this regulation (kind of current, voltages of the various parts of the installation, etc.).

Within the first 15 days of each year, the diagram and the information accompanying it, if necessary, shall be prepared by the operator and transmitted to the Engineer in Chief of Mines.

In case of important modifications or new installations, the plan and the complementary information shall be submitted to the Engineer in Chief of Mines before starting to put them in service.

Article 80

Locomotive haulage to the interior of the mine and electric haulage are not allowed unless in conformity with instructions, approved by the Engineer in Chief of Mines, governing the conditions of travel of trains and of their personnel.

Article 82

Every hoist installed outside or inside should be provided:

1. With a brake capable of stopping its operation at any position of the hoist and which can function during its operation as well as when stationary, especially in the event of breaking of pipes in fluid motors or interruption to the electric current. The brake shall be capable of being operated by the mechanician immediately and directly from his place of control.

FUTURE PROSPECTS FOR ELECTRICITY IN FRENCH MINES

In a paper delivered during April 1927 before the Society of French Electrical Engineers, M. Etienne Audibert, Engineer in Chief of Mines and Director of the Testing Station of the Central Committee of the Coal Mines of France has given an excellent résumé covering the conditions of electricity in French coal mines and showing in a general way the probable attitude toward the future use of electricity in gassy mines. This paper is too long and comprehensive to quote from at length, but M. Audibert's conclusions which are very interesting are quoted in full in the following:

I will now conclude in this fashion: There is in existence to-day electrical equipment which may rightly be termed gas-proof. The miners admit its worth, but an unfortunate experience has taught them that the safety is only relative, that certainly apparatus of this type reduce the probability of causing a fire-damp explosion to the minimum but still do

not entirely eliminate it. Look at our flame lamps; gentlemen, for almost a century, hundreds of thousands of workmen have used them in gassy workings; the number of accidents they have caused is infinitesimal in comparison with the number of shifts they have been used. But, still it is not absolutely nil.... The same thing will happen with explosion-proof electrical apparatus. But the least we can do is to surround with all possible precautions the introduction into our working places of equipment which, possibly, might be considered a new cause of fire-damp ignitions. One brief formula appears to me to be able to define the principle of the precautions we must take. It consists in saying that the risk inherent in the use of "safety" electrical equipment may readily be chanced in all cases where the apparatus is installed in working places having a constant air current of appreciable velocity. Therefore we may put motors in haulageways without any hesitation; we can try installing them, but with very much more hesitancy, at working faces or in their immediate vicinity. However, it will be very easy for us to decide, since for French mines, at least, places where electricity can be used to advantage are not often met with.

I think these explanations have made our point of view obvious to you and that you will no longer feel dubious in regard to our needs of electrical equipment. To avoid misunderstanding, I still feel that I should add a further word:

Electricity, as compared with compressed air, has a great advantage in the matter of cost. In a new mine, therefore, it is obvious that one would prefer that its use would be possible every time. The question of its introduction into a mine already equipped with compressed air may possibly appear in a different light, which will vary according to circumstances, depending on whether the compressed-air equipment is already more or less deteriorated and whether a more or less large percentage of it should be kept in service.

In a general way - and with this I will conclude - I ask you, gentlemen, to remember that you will always find the mine operators anxious to study, in collaboration with you, the problems which arise in practice in their workings in regard to the apparatus which you may see fit to design for their use.

VISIT TO FRENCH TESTING STATIONS

On September 10, 1927, accompanied by G. Allsop of the British Safety in Mines Research Station and J. A. B. Horsley, H. M. Electrical Inspector of Mines, Great Britain, the writer had the pleasure of meeting M. Etienne Audibert, Chief Engineer of Mines and Director of the Testing Station of the Central Committee of the Coal Mines of France, at Senlis, his headquarters. Senlis is a small town about one hour's distance by rail from Paris. Director Audibert proved a most genial host to the party, and after discussing mine-safety work, including certain phases of electrical work, made arrangements for a visit to Montlucon on September 12.

Accordingly, on September 12 the same party arrived at Montlucon, about 200 miles south of Paris, where they were met by M. Delmas, M. Audibert's assistant, who treated them with considerate courtesy during their visit to the testing station.

The testing work at Montlucon is carried on at two points not far apart. Research work on pulverized coal is being conducted at one point. The building in which this work was being done was large and roomy and would permit a considerable extension of their work if circumstances should require expansion. The chief item of interest at the other testing station was the facilities for testing electrical equipment, which excelled any other seen in the four countries visited; in fact, the electrical testing equipment was to all purpose a duplicate of the Gallery No. 5 outfit used for testing electrical equipment at the Pittsburgh Experiment Station of the Bureau of Mines. It is understood that the French engineers visited other European testing stations and studied other testing outfits in England, Germany, and Belgium before deciding on duplicating the American outfit. Owing to not having working drawings of the gallery, the draftsman making up the plans for the French outfit apparently made a mistake as to size of shields for the four observation windows, which have been constructed so small as to render the observation of the actual test somewhat dangerous to the observer.

Testing for Permissibility

At the time of the visit about 35 applications for tests of electrical accessories had been received, and six approvals had been granted. This approval work was for the use of apparatus in potash mines, as electrical-motored equipment had not been admitted to gassy coal mines. At that time both Director Audibert and his assistant, M. Delmas, were hopeful that regulations would soon be issued which would permit the introduction of permissible motors into gassy French mines.

TENTATIVE RULES COVERING TESTS OF ELECTRICAL EQUIPMENT FOR GASSY MINES

It is understood that tentative rules have been prepared covering the requirements of electrical equipment for use in gassy mines and that these regulations are being circulated throughout the industry for criticism.

INVESTIGATION OF ELECTRIC LAMPS

No rules have been published with respect to the requirements for electrical lamps, but M. Delmas in a recent letter has advised that the Commission of Fire Damp considers the following items with respect to construction of lamps:

1. That the lamp is strongly and ruggedly constructed (using "ruggedly" in the ordinary sense).

2. That it is suitably protected against bumps (that is, that the glass guard for the bulb is thick enough--4 or 5 millimeters (0.157 or 0.197 inches)--; that it is covered by a top which projects far enough that it is sufficiently protected against impacts).

3. That the interior of the lamp is sufficiently insulated from the external atmosphere. The approved lamps have three threads in continuous engagement in the locked position. Several lamps have been rejected because the upper threads of the jar were broken by the nick in which the bolt of the magnetic lock falls.

4. That connection is made by screwing and unscrewing and not by means of external parts.

5. Finally, for some months, the Fire Damp Commission requires that the lamps should be furnished with a device assuring instantaneous cutting out of the circuit to the filament in case the protecting glass of the bulb is broken.

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

HAZARDS IN THE USE OF DELAY-ACTION DETONATORS IN COAL MINES

BY

D. HARRINGTON AND S. P. HOWELL

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

HAZARDS IN THE USE OF DELAY-ACTION DETONATORS IN COAL MINES [1]

By D. Harrington[2] and S. P. Howell[3]

Recently the attention of the U. S. Bureau of Mines has in a number of instances been called to the firing of shots of permissible explosives in coal mines with delay-action detonators. In one bituminous coal mine they were used to bring down the coal for conveyor loading; in another in blasting pillars, which were not undercut; in another in one of the two charges separated by stemming in deep drill holes where the coal was undercut and sheared and mechanically loaded; in another in shooting top coal; and in another in the shooting of coal faces electrically from the surface. Some of these mines were rated as gassy by the State Mining Department in the State concerned. In most instances the charging of holes was done while the working shift was in the mine, and in some the shots were blasted while the working shift was in the mine.

In all of these cases the shots were dependent - that is, the efficacy of the succeeding shots was dependent - upon the firing of the preceding shot or shots. The first shot, or possibly two or three shots, was usually fired with a no-delay electric detonator or an instantaneous electric detonator.

The hazards attending the firing of such shots are obvious when one considers that the first shot or shots may release inflammable gas, will usually produce and put into suspension more or less fine coal-dust, and unquestionably on occasion will so bring down the coal that there is an inadequate burden on succeeding holes. In some cases this burden may be but a few inches, or the shot may be entirely exposed. When the succeeding shots go off, the flame of the explosive will be likely to ignite the gas or the coal-dust and thereby produce a local gas or dust explosion which may or may not be followed by a widespread dust explosion, depending upon the surrounding conditions. Should a first or second delay shot fail to detonate, the burden for succeeding shots may be so excessive that these shots will "blow out;" this also produces conditions which may result

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2 Chief engineer, safety division, U. S. Bureau of Mines.
3 Explosives engineer, U. S. Bureau of Mines, Pittsburgh Experiment Station, Pittsburgh, Pa.

in ignition of gas or dust or of both, even when permissible explosives are used. As a matter of fact permissible explosives used in delay-detonator shooting lose their permissibility, since there is a possibility, even a probability, that in some of the shots the explosive may be detonated essentially in the open; under such circumstances there may be ignition of gas or dust or of both, and, moreover, the first shots may cause later ones to misfire or even be mixed into the coal pile in an undetonated condition later on to constitute a hazard to workers or to consumers of coal.

Perhaps it should be stated that delay-action electric detonators are very desirable blasting accessories to use in mines other than coal mines, in quarries, and in general blasting where there is no gas or dust hazard. In fact, their use is increasing for these purposes. However, careful coal-mining people have long recognized the numerous dangers of dependent shots in any kind of coal-mine blasting. Some States absolutely prohibit the use of dependent shots in coal mines, and all States should prohibit their use. There is absolutely no question that blasting with delay action detonators is dependent shooting.

There is no coal-mine blasting problem which can not be met successfully by the use of permissible explosives properly confined in drill holes and fired by instantaneous electric detonators; certainly delay-action electric detonators should not be used in any coal mine when any person is in the mine during the blasting period. If an operating company for any reason insists upon using delay-action detonators in any kind of blasting in a coal mine, the blasting should be done when all persons, including the shot firers, are out of the mine.

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CIRCULAR 6148 JUNE, 1929

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE -- BUREAU OF MINES

SELECTED BIBLIOGRAPHY OF MINERALS AND THEIR IDENTIFICATION

BY

OLIVER BOWLES

Circular 6148
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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SELECTED BIBLIOGRAPHY OF MINERALS AND THEIR IDENTIFICATION [1]

By Oliver Bowles[2]

INTRODUCTION

Many inquires are received by the United States Bureau of Mines for the names of elementary books on geology, mineralogy, methods of identifying minerals, prospecting, and similar subjects. In response to this demand the following brief bibliography has been prepared. As many of the inquiries are received from those who have limited technical knowledge of the subjects involved, the bibliography includes the simpler texts which present the subjects in nontechnical language. Other texts contain glossaries which define the technical terms used. A short note following the title indicates the character of each book, the number of pages, and the price. Thus, elementary mineralogists or geologists, prospectors, mineral collectors, nature students, or travellers may select the texts that best suit their requirements and their capabilities.

To supply the needs of more advanced students, quite a number of the standard texts used in schools and colleges are included in a second group. A short list of books on economic geology and mineralogy has also been added.

ELEMENTARY BOOKS

The following books are elementary in character, and are best adapted for those who have a limited technical knowledge of geology and mineralogy:

Anderson, J. W. Prospector's handbook. 12th rev. ed., D. Van Nostrand Co., Inc., New York, 210 pp. $2. A guide for the prospector and traveler in search of metal-bearing or other valuable minerals. Contains a glossary of terms used.

Burdick, A. J. Valuable minerals, how to find and know them. 2nd ed., The Beaumont Gazette, Beaumont, Cal., 1928. 32 pp. 50 cents. A nontechnical pamphlet consisting of notes on prospecting and mineral testing.

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2 Supervising engineer, building materials section, U. S. Bureau of Mines.

Butler, G. M. A pocket handbook of minerals. 2nd ed., John Wiley & Sons, Inc., New York. 311 pp. $3. A book designed for use in the field or classroom; contains little reference to chemical tests. Gives physical characters needed to identify most of the minerals which students or collectors are apt to encounter.

Cox, H. S. Prospecting for minerals. 8th ed., J. B. Lippincott Co., Philadelphia, 1921, 260 pp. $2.50. Contains brief notes on geology, descriptions of minerals, determinative tables, and a discussion of non-metallic minerals, ores, and fuels. Written in nontechnical language, easily understood by beginners.

Dana, E. S. Minerals and how to study them. 2nd ed., John Wiley & Sons, Inc., New York, 1897, 380 pp. $2.

Dana, E. S., and Ford, W. E. Dana's Manual of Mineralogy. 13th ed., John Wiley & Sons, Inc., New York, 1912, 460 pp. $3. A book for the student of elementary mineralogy, the mining engineer, the geologist, the prospector, and the collector. Revised and rewritten by W. E. Ford.

Foye, J. C. Handbook of mineralogy. 5th rev. ed., No. 86, Van Nostrand Science Series. D. Van Nostrand Co., Inc., New York. 75 cents. Covers determination, description, and classification of common minerals.

Gratacap, L. P. A popular guide to mineral collections. D. Van Nostrand Co., Inc., New York, 335 pp. $4. Prepared for the use of visitors to public cabinets of minerals, and for elementary teaching in mineralogy. (Illustrated.)

Loomis, F. B. Field-book of common rocks and minerals. G. P. Putnam's Sons, New York, 1923. $3.50. Designed for the identification of rocks and minerals. Contains 47 colored plates and numerous illustrations from photographs.

McLeod, Alexander. Practical instructions in the search for, and determination of the useful minerals, including the rare ores. 2nd. ed., John Wiley & Sons, Inc., New York, 254 pp. $2.50. Furnishes simple means for determination of minerals.

Merritt, W. H. Field testing for gold and silver. D. Van Nostrand Co., Inc., New York, 155 pp. $2.50. A practical manual for prospectors and miners.

Miller, W. G., and Parsons, A. L. Minerals and how they occur. Rev. ed., The Copp Clark Co., Ltd., Toronto, 1928, 255 pp. $1.25. An outdoor book to meet the needs of the general reader and prospector, as well as those of students in the secondary schools. Describes rocks, fossils, crystals, and minerals in a very simple way.

Osborn, H. S. Prospector's field-book and guide. Revised by M. W. von Bernewitz. 10th ed., Henry Carey Baird & Co., Inc., New York, 364 pp. $3. Describes minerals and their occurrence with methods of testing. Contains useful tables and a glossary of terms.

Platt, William. A popular geology. The MacMillan Co., New York, 1924, 118 pp. $1. A very simple popular discussion of soils, rocks, fossils, and mountains.

Scott, W. B. An introduction to geology. 2nd ed., The Macmillan Co., New York, 1907, 816 pp. $4. A book intended to serve as an introduction to the science of geology, both for the future specialist, and for those who wish to gain a general knowledge of the science.

von Bernewitz, M. W. Handbook for prospectors. McGraw-Hill Book Co., Inc., New York, 319 pp. $3. A guidebook for prospectors, giving practical information on equipment, methods of procedure, and mining laws. Contains brief reference to geology, mineralogy, and the occurrence, description, detection, and use of various minerals.

STANDARD TEXTBOOKS

The following are standard textbooks on geology and mineralogy:

Brush, G. J. Manual of determinative mineralogy, with an introduction on blowpipe analysis. Revised and enlarged by S. L. Penfield. 16th ed., John Wiley & Sons, Inc., New York, 1909, 312 pp. $3.50. A standard textbook on mineralogy and blowpipe analysis.

Cahen, Edward, and Wootton, W. O. Mineralogy of the rarer metals. 2nd ed., J. P. Lippincott Co., Philadelphia, 1920, 246 pp. $6. Presents a discussion of all the rare metals, under such headings as detection, properties, metallurgy, industrial application, production, and value.

Dana, E. S. A textbook of mineralogy, with an extended treatise on crystallography and physical mineralogy. Revised and enlarged by W. E. Ford. 3rd ed., John Wiley & Sons, Inc., New York, 1922, 720 pp. $5. A comprehensive treatment of crystallography and mineralogy.

Eakle, A. S. Mineral tables for the determination of minerals by their physical properties. John Wiley & Sons, Inc., New York, 1922, 73 pp. $1.50.

Kemp, J. F. Handbook of rocks for use without the microscope. 5th rev. ed., D. Van Nostrand Co., New York, 283 pp. $3. A standard textbook and guide in the field classification of rocks; for students, mining men, and geologists. Contains a glossary of the names of rocks and other lithological terms.

Kraus, E. H., and Hunt, W. F. Tables for the determination of minerals by means of their physical properties, occurrences, and associates. McGraw-Hill Book Co., Inc., New York, 1911, 254 pp. $2.50.

______________. Mineralogy. 2nd ed., McGraw-Hill Book Co., New York, 1928, 604 pp. $5. A general mineralogy designed particularly for classes of beginning students. It has numerous illustrations, including photographs of minerals and crystals; also gives data on gems and precious stones, and on uses of economic minerals.

Lewis, J. Volney. A manual of determinative mineralogy. 3rd. ed., John Wiley & Sons, Inc., New York, 1921, 298 pp. $3. A standard textbook containing tables for the determination of minerals by means of their physical properties, also by blowpipe and chemical tests.

Moses, A. J., and Parsons, C. L. Elements of mineralogy, crystallography, and blowpipe analysis from a practical standpoint. 5th ed., D. Van Nostrand Co., New York, 1916, 631 pp. $4.50. A standard reference book including descriptions of minerals, their formation and occurrence, tests for their identification, and their economic importance and uses in the arts.

Phillips, A. H. Mineralogy, an introduction to the theoretical and practical study of minerals. The Macmillan Co., New York, 1912, 699 pp. $4.50. A standard textbook of mineralogy, in three parts: I, Crystallography; II, Descriptive Mineralogy; III, Determinative Mineralogy.

Pirsson, L. V., and Knopf, Adolph. Rocks and rock minerals. 2nd. ed., John Wiley & Sons, Inc., New York, 1926, 426 pp. $3.50. A standard textbook, with a particularly instructive table for determination of common rocks.
Rogers, A. F. Introduction to the study of minerals and rocks. McGraw-Hill Book Co., New York, 1921, 527 pp. $4. Covers the whole field of mineralogy, including crystallography, blowpipe analysis, and descriptive and determinative mineralogy; for use in the field or in the classroom.
Rutley, Frank. Elements of mineralogy. 19th ed., revised and enlarged. D. Van Nostrand Co., Inc., New York, $2.50. A general text covering all common minerals.
Warren, C. H. A manual of determinative mineralogy. McGraw-Hill Book Co., New York, 1922, 163 pp. $2. A book for use in general courses in mineralogy. Contains simple blowpipe tests and determinative tables. (Flexible, pocket size.)

STANDARD ECONOMIC TEXTS

The following books are standard texts on economic geology and mineralogy:

Emmons, W. H. The principles of economic geology. McGraw-Hill Book Co., Inc., New York, 1918, 612 pp. $5. The material relating to metallic ores and minerals is particularly valuable.
Ladoo, R. B. Non-Metallic Minerals. McGraw-Hill Book Co., New York, 1925, 686 pp. $6. Covers the whole field of nonmetallic minerals except fuels and hydrocarbons, emphasizing particularly the methods of mining and preparation, uses, markets, specifications, and tests.
Lindgren, Waldemar. Mineral deposits. 3rd ed., McGraw-Hill Book Co., Inc., New York, 1928, 1049 pp. $7. A leading treatise on economic geology.
Ries, Heinrich. Economic geology. 5th ed., John Wiley & Sons, Inc., New York, 843 pp. $5. Contains material relating to clays and other nonmetallic minerals which is particularly valuable.

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